I. C. S.
COAL MINER'S
HANDBOOK

The Coal Miner's Handbook

A HANDY REFERENCE BOOK

FOR

Coal Miners, Pit Bosses, Fire Bosses, Foremen, Superintendents, Managers, Engineers, and All Persons Interested in the Subject of Coal Mining

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BY

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PREFACE

This Handbook is intended for all who are interested in coal mining and for all who are employed in and about the coal mines. While the treatment of some of the subjects included is necessarily brief, we have striven to anticipate the daily wants of the user and to give him, in the manner best suited to his needs, the information he desires. The breaker boy, the driver, the helper will find many useful hints to help him in his work and to assist him in securing advancement.

From a vast number of reference tables and formulas only the best have been selected and incorporated, and these have been thoroughly explained. This feature in itself should result in a great saving of time and in preventing the selection of the wrong table or formula. The subjects of surveying, use and care of wire ropes in connection with hoisting and haulage, electricity, opening of mines, timbering, methods of working, and of ventilation have received special attention. Safety appliances, which include electric signaling devices and safety lamps, are treated in detail, as is also the care and use of

explosives. Considerable space is also devoted to the treatment of persons injured in and about the mines and those overcome with mine gases.

The man employed on the surface will find recorded many useful facts dealing with surface plants, considerable space being devoted to dams, pumping, steam, preparation of coal, etc.

This little book should satisfy a want long existing in the coal-mining industry for a ready pocket reference containing information that anyonce can use in making calculations and settling questions.

International Correspondence Schools

March 1, 1913

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The Coal Miner's Handbook

USEFUL TABLES

WEIGHTS AND MEASURES

LINEAR MEASURE

12	inches (in.)	= 1 foot	ft.
3	feet	= 1 yard	yd.
$5\frac{1}{2}$	yards	= 1 rod	rd.
40	rods	= 1 furlong	fur.
8	furlongs	= 1 mile	mi.

in.	*ft.*	*yd.*	*rd.*	*fur.*	*mi.*
36 =	3 =	1			
198 =	16.5 =	5.5 =	1		
7,920 =	660 =	220 =	40 =	1	
63,360 =	5,280 =	1,760 =	320 =	8 =	1

SQUARE MEASURE

144	square inches (sq. in.)	= 1 square foot	sq. ft.
9	square feet	= 1 square yard	sq. yd.
$30\frac{1}{4}$	square yards	= 1 square rod	sq. rd.
160	square rods	= 1 acre	A.
640	acres	= 1 square mile	sq. mi.

sq. mi.	*A.*	*sq. rd.*	*sq. yd.*	*sq. ft.*	*sq. in.*
1 =	640 =	102,400 =	3,097,600 =	27,878,400 =	4,014,489,600

MEASURE OF ANGLES OR ARCS

60 seconds (″)	= 1 minute	′
60 minutes	= 1 degree	°
90 degrees	= 1 rt. angle or quadrant	□
360 degrees	= 1 circle	cir.

1 cir. = 360° = 21,600′ = 1,296,000″

CUBIC MEASURE

1,728 cubic inches (cu. in.)	= 1 cubic foot	cu. ft.
27 cubic feet	= 1 cubic yard	cu. yd.
128 cubic feet	= 1 cord	cd.
$24\frac{3}{4}$ cubic feet	= 1 perch	P.

cu. yd. *cu. ft.* *cu. in.*
1 = 27 = 46,656

AVOIRDUPOIS WEIGHT

$437\frac{1}{2}$ grains (gr.)	= 1 ounce	oz.
16 ounces	= 1 pound	lb.
100 pounds	= 1 hundredweight	cwt.
20 cwt., or 2,000 lb.	= 1 ton	T.

T. *cwt.* *lb.* *oz.* *gr.*
1 = 20 = 2,000 = 32,000 = 14,000,000

The avoirdupois pound contains 7,000 gr.

LONG-TON TABLE

16 ounces	= 1 pound	lb.
112 pounds	= 1 hundredweight	cwt.
20 cwt., or 2,240 lb.	= 1 ton	T.

DRY MEASURE

2 pints (pt.)	= 1 quart	qt.
8 quarts	= 1 peck	pk.
4 pecks	= 1 bushel	bu.

bu. *pk.* *qt.* *pt.*
1 = 4 = 32 = 64

The *U. S. struck bushel* contains 2,150.42 cu. in. = 1.2444 cu. ft. By law, its dimensions are those of a cylinder $18\frac{1}{2}$ in. in diameter and 8 in. deep. The heaped bushel is equal to $1\frac{1}{4}$ struck bu., the cone being 6 in. high. For approximations,

the bushel may be taken at $1\frac{1}{4}$ cu. ft. or 1 cu. ft. may be considered $\frac{4}{5}$ bu.

The *British bushel* contains 2,218.19 cu. in. = 1.2837 cu. ft. = 1.032 U. S. bu.

The *dry gallon* contains 268.8 cu. in., or $\frac{1}{8}$ struck bu.

LIQUID MEASURE

4	gills (gi.)	= 1 pint	pt.
2	pints	= 1 quart	qt.
4	quarts	= 1 gallon	gal.
$31\frac{1}{2}$	gallons	= 1 barrel	bbl.
2	barrels, or 63 gallons	= 1 hogshead	hhd.

hhd.		*bbl.*		*gal.*		*qt.*		*pt.*		*gi.*
1	=	2	=	63	=	252	=	504	=	2,016

The *U. S. gallon* contains 231 cu. in. = .134 cu. ft., nearly; or 1 cu. ft. contains 7.481 gal. The following cylinders contain the given measures very closely:

	Diam. Inches	Height Inches		Diam. Inches	Height Inches
Gill	$1\frac{3}{4}$	3	Gallon	7	6
Pint	$3\frac{1}{2}$	3	8 gal.	14	12
Quart	$3\frac{1}{2}$	6	10 gal.	14	15

When water is at its maximum density, 1 cu. ft. weighs 62.425 lb. and 1 gal. weighs 8.345 lb.

For approximations, 1 cu. ft. of water is considered equal to $7\frac{1}{2}$ gal., and 1 gal. as weighing $8\frac{1}{2}$ lb.

The *British imperial gallon*, both liquid and dry, contains 277.274 cu. in. = .16046 cu. ft., and is equivalent to the volume of 10 lb. of pure water at 62° F.

To reduce British to U. S. liquid gallons, multiply by 1.2. Conversely, to convert U. S. into British liquid gallons, divide by 1.2; or, increase the number of gallons one-fifth.

METRIC SYSTEM

The *metric system* is based on the *meter*, which, according to the U. S. Coast and Geodetic Survey report of 1884, is equal to 39.370432 in. The value commonly used is 39.37 in., and is authorized by the U. S. government. The meter is defined

as one ten-millionth the distance from the pole to the equator measured on a meridian passing near Paris.

There are three principal units—the meter, the liter (pronounced lee-ter), and the gram, the units of length, capacity, and weight, respectively. Multiples of these units are obtained by prefixing to the names of the principal units the Greek words *deca* (10), *hecto* (100), and *kilo* (1,000); the submultiples, or divisions, are obtained by prefixing the Latin words *deci* ($\frac{1}{10}$), *centi* ($\frac{1}{100}$), and *milli* ($\frac{1}{1000}$). These prefixes form the key to the entire system. In the following tables, the abbreviations of the principal units of these submultiples begin with a small letter, while those of the multiples begin with a capital letter; they should always be written as here printed.

MEASURE OF LENGTH

10 millimeters (mm.)	= 1 centimeter	cm.
10 centimeters	= 1 decimeter	dm.
10 decimeters	= 1 meter	m.
10 meters	= 1 decameter	Dm.
10 decameters	= 1 hectometer	Hm.
10 hectometers	= 1 kilometer	Km.

MEASURES OF SURFACE (NOT LAND)

100 square millimeters (sq. mm.)	= 1 square centimeter	sq. cm.
100 square centimeters	= 1 square decimeter	sq. dm.
100 square decimeters	= 1 square meter	sq. m.

MEASURES OF VOLUME

1,000 cubic millimeters (cu. mm.)	= 1 cubic centimeter	c. c. or cu. cm.
1,000 cubic centimeters	= 1 cubic decimeter	cu. dm.
1,000 cubic decimeters	= 1 cubic meter	cu. m.

MEASURES OF CAPACITY

10 milliliters (ml.)	= 1 centiliter	cl.
10 centiliters	= 1 deciliter	dl.
10 deciliters	= 1 liter	l.
10 liters	= 1 decaliter	Dl.
10 decaliters	= 1 hecoliters	Hl.
10 hecoliters	= 1 kiloliters	Kl.

The *liter* is equal to the volume occupied by 1 cu. dm.

MEASURES OF WEIGHT

10 milligrams (mg.)	= 1 centigram	cg.
10 centigrams	= 1 decigram	dg.
10 decigrams	= 1 gram	g.
10 grams	= 1 decagram	Dg.
10 decagrams	= 1 hectogram	Hg.
10 hectograms	= 1 kilogram	Kg.
1,000 kilograms	= 1 ton	T.

The *gram* is the weight of 1 c. c. of pure distilled water at a temperature of 39.2° F.; the *kilogram* is the weight of 1 l. of water; the *ton* is the weight of 1 cu. m. of water.

CONVERSION TABLES

By means of the accompanying tables, metric measures can be converted into English and vice versa, by simple addition. All the figures of the values given are not required except in very exact calculations; as a rule, 4 or 5 digits only are used. To change 6,471.8 ft. into meters, consider 6,471.8 as 6,000+400+70+1+.8; also, 6,000 = 1,000×6; 400 = 100×4, etc. Hence, looking in the first column of the table entitled English Measures Into Metric, for 6 (the first figure of the given number), opposite it in the column headed Feet to Meters, is found the number 1.8287838. Using but five digits and increasing the fifth digit by 1 (as the next is greater than 5), gives 1.8288. In other words, 6 ft. = 1.8288 m.; hence, 6,000 ft. = 1,000×1.8288 = 1,828.8, simply moving the decimal point three places to the right. Likewise, it is found that 400 ft. = 121.92 m.; 70 ft. = 21.336 m.; 1 ft. = .3048 m., and .8 ft. = .2438 m. Adding as shown, gives 1,972.6046 m. as the value of 6,471.8 ft.

$$\begin{array}{r} 1{,}828.8 \\ 121.92 \\ 21.336 \\ .3048 \\ .2438 \\ \hline 1{,}972.6046 \end{array}$$

As another example, convert 19.635 kg. into pounds. Working according to the explanation just given, it is found that 19.635 kg. = 43.2875 lb.

$$\begin{array}{r} 22.046 \\ 19.8416 \\ 1.3228 \\ .0661 \\ .0110 \\ \hline 43.2875 \end{array}$$

The only difficulty in applying these tables lies in locating the decimal point; it may always be found thus: If the figure considered lies to the left of the decimal point, count each figure in order, beginning with units (but calling units' place zero), until the

desired figure is reached, then move the decimal point to the *right* as many places as the figure being considered is to the

CONVERSION TABLE

English Measures Into Metric

English	Metric: Inches to Meters	Metric: Feet to Meters	Metric: Pounds to Kilos	Metric: Gallons to Liters
1	.0253998	.3047973	.4535925	3.7853122
2	.0507996	.6095946	.9071850	7.5706244
3	.0761993	.9143919	1.3607775	11.3559366
4	.1015991	1.2191892	1.8143700	15.1412488
5	.1269989	1.5239865	2.2679625	18.9265610
6	.1523987	1.8287838	2.7215550	22.7118732
7	.1777984	2.1335811	3.1751475	26.4971854
8	.2031982	2.4383784	3.6287400	30.2824976
9	.2285980	2.7431757	4.0823325	34.0678098
10	.2539978	3.0479730	4.5359250	37.8531220

English	Metric: Square Inches to Square Meters	Metric: Square Feet to Square Meters	Metric: Cubic Feet to Cubic Meters	Metric: Pounds per Square Inch to Kilo per Square Meter
1	.000645150	.092901394	.028316094	703.08241
2	.001290300	.185802788	.056632188	1,406.16482
3	.001935450	.278704182	.084948282	2,109.24723
4	.002580600	.371605576	.113264376	2,812.32964
5	.003225750	.464506970	.141580470	3,515.41205
6	.003870900	.557408364	.169896564	4,218.49446
7	.004516050	.650309758	.198212658	4,921.57687
8	.005161200	.743211152	.226528752	5,624.65928
9	.005806350	.836112546	.254844846	6,327.74169
10	.006451500	.929013940	.283160940	7,030.82410

left of the unit figure. Thus, in the first example, 6 lies three places to the left of 1, which is in units' place; hence, the decimal point is moved three places to the right. By exchanging

the words right and left, the statement will also apply to decimals. Thus, in the sccond case above, the 5 lies three places

CONVERSION TABLE

METRIC MEASURES INTO ENGLISH

Metric	English Meters to Inches	English Meters to Feet	English Kilos to Pounds	English Liters to Gallons
1	39.370432	3.2808693	2.2046223	.2641790
2	78.740864	6.5617386	4.4092447	.5283580
3	118.111296	9.8426079	6.6138670	.7925371
4	157.481728	13.1234772	8.8184894	1.0567161
5	196.852160	16.4043465	11.0231117	1.3208951
6	236.222592	19.6852158	13.2277340	1.5850741
7	275.593024	22.9660851	15.4323564	1.8492531
8	314.963456	26.2469544	17.6369787	2.1134322
9	354.333888	29.5278237	19.8416011	2.3776112
10	393.704320	32.8086930	22.0462234	2.6417902

Metric	English Square Mcters to Square Inches	English Square Meters to Square Feet	English Cubic Meters to Cubic Feet	English Kilos per Square Meter to Pounds pcr Squarc Inch
1	1,550.03092	10.7641034	35.3156163	.001422310
2	3,100.06184	21.5282068	70.6312326	.002844620
3	4,650.09276	32.2923102	105.9468489	.004266930
4	6,200.12368	43.0564136	141.2624652	.005689240
5	7,750.15460	53.8205170	176.5780815	.007111550
6	9,300.18552	64.5846204	211.8936978	.008533860
7	10,850.21644	75.3487238	247.2093141	.009956170
8	12,400.24736	86.1128272	282.5249304	.011378480
9	13,950.27828	96.8769306	317.8405467	.012800790
10	15,500.30920	107.6410340	353.1561630	.014223100

to the right of units' place; hence, the decimal point in the number taken from the table is moved three places to the left.

TIMBER AND BOARD MEASURE

TIMBER MEASURE

Volume of Round Timber.—The volume of round timber, in cubic feet, equals the length multiplied by one-fourth the product of mean girth and diameter, all dimensions being in feet. If length is given in feet and girth and diameter in inches, divide by 144; if all dimensions are in inches, divide by 1,728.

AREA OF ROUND TIMBER

¼ Girths Inches	Area Square Feet	¼ Girths Inches	Area Square Feet	¼ Girths Inches	Area Square Feet
6	.250	12¼	1.04	19	2.50
6¼	.272	12½	1.08	19½	2.64
6½	.294	12¾	1.12	20	2.77
6¾	.317	13	1.17	20½	2.91
7	.340	13¼	1.21	21	3.06
7¼	.364	13½	1.26	21½	3.20
7½	.390	13¾	1.31	22	3.36
7¾	.417	14	1.36	22½	3.51
8	.444	14¼	1.41	23	3.67
8¼	.472	14½	1.46	23½	3.83
8½	.501	14¾	1.51	24	4.00
8¾	.531	15	1.56	24½	4.16
9	.562	15¼	1.61	25	4.34
9¼	.594	15½	1.66	25½	4.51
9½	.626	15¾	1.72	26	4.69
9¾	.659	16	1.77	26½	4.87
10	.694	16¼	1.83	27	5.06
10¼	.730	16½	1.89	27½	5.25
10½	.766	16¾	1.94	28	5.44
10¾	.803	17	2.00	28½	5.64
11	.840	17¼	2.09	29	5.84
11¼	.878	17½	2.12	29½	6.04
11½	.918	17¾	2.18	30	6.25
11¾	.959	18	2.25		
12	1.000	18½	2.37		

In the accompanying table is given the area of round timber. The *area* corresponding to ¼ girth (mean), in inches, multiplied by the length, in feet, equals the solidity, in feet and decimal parts.

AREA OF CUT TIMBER

Breadth Inches	Area of 1 Lin. Ft.	Breadth Inches	Area of 1 Lin. Ft.	Breadth Inches	Area of 1 Lin. Ft.
$\frac{1}{4}$	.021	$4\frac{1}{4}$	.354	$8\frac{1}{4}$	.688
$\frac{1}{2}$	.042	$4\frac{1}{2}$	.375	$8\frac{1}{2}$	.708
$\frac{3}{4}$	.063	$4\frac{3}{4}$	.396	$8\frac{3}{4}$	.729
1	.083	5	.417	9	.750
$1\frac{1}{4}$	.104	$5\frac{1}{4}$	.438	$9\frac{1}{4}$	.771
$1\frac{1}{2}$	.125	$5\frac{1}{2}$	.458	$9\frac{1}{2}$	.792
$1\frac{3}{4}$	.146	$5\frac{3}{4}$	.479	$9\frac{3}{4}$	.813
2	.167	6	.500	10	.833
$2\frac{1}{4}$	.188	$6\frac{1}{4}$	.521	$10\frac{1}{4}$	.854
$2\frac{1}{2}$	.208	$6\frac{1}{2}$	.542	$10\frac{1}{2}$	.875
$2\frac{3}{4}$	.229	$6\frac{3}{4}$	.563	$10\frac{3}{4}$	.896
3	.250	7	.583	11	.917
$3\frac{1}{4}$	.271	$7\frac{1}{4}$	.604	$11\frac{1}{4}$	.938
$3\frac{1}{2}$	.292	$7\frac{1}{2}$	.625	$11\frac{1}{2}$	.958
$3\frac{3}{4}$	.313	$7\frac{3}{4}$	.646	$11\frac{3}{4}$	.979
4	.333	8	.667	12	1.000

DECIMAL EQUIVALENTS OF PARTS OF 1 IN.

Part of Inch	Equivalent	Part of Inch	Equivalent	Part of Inch	Equivalent	Part of Inch	Equivalent
$\frac{1}{64}$	.015625	$\frac{17}{64}$	.265625	$\frac{33}{64}$	.515625	$\frac{49}{64}$	.765625
$\frac{1}{32}$	.031250	$\frac{9}{32}$	.281250	$\frac{17}{32}$	.531250	$\frac{25}{32}$	.781250
$\frac{3}{64}$	.046875	$\frac{19}{64}$	.296875	$\frac{35}{64}$	.546875	$\frac{51}{64}$	.796875
$\frac{1}{16}$	.062500	$\frac{5}{16}$	.312500	$\frac{9}{16}$	.562500	$\frac{13}{16}$	.812500
$\frac{5}{64}$	.078125	$\frac{21}{64}$	.328125	$\frac{37}{64}$	.578125	$\frac{53}{64}$	.828125
$\frac{3}{32}$	.093750	$\frac{11}{32}$	.343750	$\frac{19}{32}$	.593750	$\frac{27}{32}$	.843750
$\frac{7}{64}$	.109375	$\frac{23}{64}$	.359375	$\frac{39}{64}$	.609375	$\frac{55}{64}$	.859375
$\frac{1}{8}$	.125000	$\frac{3}{8}$	.375000	$\frac{5}{8}$	.625000	$\frac{7}{8}$	.875000
$\frac{9}{64}$	.140625	$\frac{25}{64}$	.390625	$\frac{41}{64}$	.640625	$\frac{57}{64}$	.890625
$\frac{5}{32}$	.156250	$\frac{13}{32}$	.406250	$\frac{21}{32}$	.656250	$\frac{29}{32}$	.906250
$\frac{11}{64}$	.171875	$\frac{27}{64}$	.421875	$\frac{43}{64}$	.671875	$\frac{59}{64}$	.921875
$\frac{3}{16}$	.187500	$\frac{7}{16}$	.437500	$\frac{11}{16}$	.687500	$\frac{15}{16}$	.937500
$\frac{13}{64}$	.203125	$\frac{29}{64}$	.453125	$\frac{45}{64}$	,703125	$\frac{61}{64}$	.953125
$\frac{7}{32}$	.218750	$\frac{15}{32}$	.468750	$\frac{23}{32}$	.718750	$\frac{31}{32}$	.968750
$\frac{15}{64}$	.234375	$\frac{31}{64}$	.484375	$\frac{47}{64}$	.734375	$\frac{63}{64}$	.984375
$\frac{1}{4}$	.250000	$\frac{1}{2}$	.500000	$\frac{3}{4}$	.750000	1	1

DECIMALS OF A FOOT FOR EACH 1-32 IN.

In.	0″	1″	2″	3″	4″	5″
0	0	.0833	.1667	.2500	.3333	.4167
1/32	.0026	.0859	.1693	.2526	.3359	.4193
1/16	.0052	.0885	.1719	.2552	.3385	.4219
3/32	.0078	.0911	.1745	.2578	.3411	.4245
1/8	.0104	.0937	.1771	.2604	.3437	.4271
5/32	.0130	.0964	.1797	.2630	.3464	.4297
3/16	.0156	.0990	.1823	.2656	.3490	.4323
7/32	.0182	.1016	.1849	.2682	.3516	.4349
1/4	.0208	.1042	.1875	.2708	.3542	.4375
9/32	.0234	.1068	.1901	.2734	.3568	.4401
5/16	.0260	.1094	.1927	.2760	.3594	.4427
11/32	.0286	.1120	.1953	.2786	.3620	.4453
3/8	.0312	.1146	.1979	.2812	.3646	.4479
13/32	.0339	.1172	.2005	.2839	.3672	.4505
7/16	.0365	.1198	.2031	.2865	.3698	.4531
15/32	.0391	.1224	.2057	.2891	.3724	.4557
1/2	.0417	.1250	.2083	.2917	.3750	.4583
17/32	.0443	.1276	.2109	.2943	.3776	.4609
9/16	.0469	.1302	.2135	.2969	.3802	.4635
19/32	.0495	.1328	.2161	.2995	.3828	.4661
5/8	.0521	.1354	.2188	.3021	.3854	.4688
21/32	.0547	.1380	.2214	.3047	.3880	.4714
11/16	.0573	.1406	.2240	.3073	.3906	.4740
23/32	.0599	.1432	.2266	.3099	.3932	.4766
3/4	.0625	.1458	.2292	.3125	.3958	.4792
25/32	.0651	.1484	.2318	.3151	.3984	.4818
13/16	.0677	.1510	.2344	.3177	.4010	.4844
27/32	.0703	.1536	.2370	.3203	.4036	.4870
7/8	.0729	.1562	.2396	.3229	.4062	.4896
29/32	.0755	.1589	.2422	.3255	.4089	.4922
15/16	.0781	.1615	.2448	.3281	.4115	.4948
31/32	.0807	.1641	.2474	.3307	.4141	.4974

Volume of Square Timber.—When all dimensions are in feet:

Rule.—*Multiply the breadth by the depth and that product by the length, and the product will give the volume, in cubic feet.*

When either of the dimensions is in inches:

Rule.—*Multiply as before and divide by 12.*

When any two of the dimensions are in inches:

Rule.—*Multiply as before and divide by 144.*

TABLE—(*Continued*)

In.	6″	7″	8″	9″	10″	11″
0	.5000	.5833	.6667	.7500	.8333	.9167
$\frac{1}{32}$	.5026	.5859	.6693	.7526	.8359	.9193
$\frac{1}{16}$	.5052	.5885	.6719	.7552	.8385	.9219
$\frac{3}{32}$	.5078	.5911	.6745	.7578	.8411	.9245
$\frac{1}{8}$	.5104	.5937	.6771	.7604	.8437	.9271
$\frac{5}{32}$	.5130	.5964	.6797	.7630	.8464	.9297
$\frac{3}{16}$	.5156	.5990	.6823	.7656	.8490	.9323
$\frac{7}{32}$	.5182	.6016	.6849	.7682	.8516	.9349
$\frac{1}{4}$	.5208	.6042	.6875	.7708	.8542	.9375
$\frac{9}{32}$	.5234	.6068	.6901	.7734	.8568	.9401
$\frac{5}{16}$	.5260	.6094	.6927	.7760	.8594	.9427
$\frac{11}{32}$	.5286	.6120	.6953	.7786	.8620	.9453
$\frac{3}{8}$	.5312	.6146	.6979	.7812	.8646	.9479
$\frac{13}{32}$	.5339	.6172	.7005	.7839	.8672	.9505
$\frac{7}{16}$	.5365	.6198	.7031	.7865	.8698	.9531
$\frac{15}{32}$	.5391	.6224	.7057	.7891	.8724	.9557
$\frac{1}{2}$	.5417	.6250	.7083	.7917	.8750	.9583
$\frac{17}{32}$	.5443	.6276	.7109	.7943	.8776	.9609
$\frac{9}{16}$	.5469	.6302	.7135	.7969	.8802	.9635
$\frac{19}{32}$	.5495	.6328	.7161	.7995	.8828	.9661
$\frac{5}{8}$	.5521	.6354	.7188	.8021	.8854	.9688
$\frac{21}{32}$	.5547	.6380	.7214	.8047	.8880	.9714
$\frac{11}{16}$	.5573	.6406	.7240	.8073	.8906	.9740
$\frac{23}{32}$	.5599	.6432	.7266	.8099	.8932	.9766
$\frac{3}{4}$	.5625	.6458	.7292	.8125	.8958	.9792
$\frac{25}{32}$	.5651	.6484	.7318	.8151	.8984	.9818
$\frac{13}{16}$	.5677	.6510	.7344	.8177	.9010	.9844
$\frac{27}{32}$	.5703	.6536	.7370	.8203	.9036	.9870
$\frac{7}{8}$	.5729	.6562	.7396	.8229	.9062	.9896
$\frac{29}{32}$	.5755	.6589	.7422	.8255	.9089	.9922
$\frac{15}{16}$	.5781	.6615	.7448	.8281	.9115	.9948
$\frac{31}{32}$	.5807	.6641	.7474	.8307	.9141	.9974

BOARD MEASURE

In measuring, boards are assumed to be 1 in. thick. The number of feet, board measure (B. M.), in a given board or stick of timber, equals the length, in feet, multiplied by the breadth, in feet, multiplied by the thickness, in inches.

Area of 1 lin. ft. multiplied by length, in feet, will give superficial contents, in square feet.

DECIMAL EQUIVALENTS

In many cases of taking measurements, it is desirable to change a fraction of an inch or foot to decimals, or getting the nearest fraction of an inch or foot from a calculation in which a large decimal appears. The preceding tables give the decimal equivalents of each $\frac{1}{64}$ in. and the decimal equivalents of 1 ft. for each $\frac{1}{32}$ in.

TRAVERSE TABLES

To use the *traverse tables*, find the number of degrees in the left-hand column if the angle is less than 45°, and in the right-hand column if greater than 45°. The numbers on the same line running across the page are the latitudes and departures for that angle and for the respective distances, 1, 2, 3, 4, 5, 6, 7, 8, 9, which appear at the top of the pages. Thus, if the bearing of a line is 10° and the distance is 4, the latitude will be 3.939 and the departure .695; with the same bearing, and the distance 8, the latitude will be 7.878 and the departure 1.389. The latitude and departure for 80 is 10 times the latitude and departure for 8, and is found by moving the decimal point one place to the right; that for 500 is 100 times the latitude and departure for 5, and is ound by moving the decimal point two places to the right and so on. By moving the decimal point one, two, or more places to the right, the latitude and departure may be found for any multiple of any number given in the table. In finding the latitude and departure for any number such as 453, the number is resolv ed into three numbers, viz. 400, 50, 3, and the latitude and departure for each is taken from the table and then added together.

Rule.—*Write down the latitude and departure, neglecting the decimal points, for the first figure of the given distance; write under them the latitude and departure for the second figure, setting them one place farther to the right; under these, place the latitude and departure for the third figure, setting them one place still farther to the right, and so continue until all the figures of the given distance have been used; add these latitudes and departures, and point off on the right of their sums a number of decimal places*

equal to the number of decimal places to which the tables being used are carried; the resulting numbers will be the latitude and departure of the given distance in feet, links, chains, or whatever unit of measurement is adopted. Should the departure or latitude consist only of a decimal, a cipher should be inserted before the decimal, as in the departures of example 1.

EXAMPLE 1.—A bearing is 16° and the distance 725 ft.; what is the latitude and departure?

SOLUTION.—Applying the rule just given:

Distances	*Latitudes*	*Departures*
700	6729	1929
20	1923	0551
5	4806	1378
725	696.936	199.788

Taking the nearest whole numbers and rejecting the decimals, the latitude and departure are 697 and 200.

When a 0 occurs in the given number, the next figure must be set *two* places to the right as in the following example:

EXAMPLE 2.—The bearing is 22° and the distance 907 ft.; required, the latitude and departure.

SOLUTION.—Applying the rule just given:

Distances	*Latitudes*	*Departures*
900	8345	3371
7	6490	2622
907	840.990	339.722

Here the place of 0 both in the distance column and in the latitude and departure columns is occupied by a dash —. Rejecting the decimals, the latitude is 841 ft. and the departure 340 ft.

When the bearing is more than 45°, the names of the columns must be read from the bottom of the page. The latitude of any bearing, as 60°, is the departure of its complement, 30°; and the departure of any bearing, as 30°, is the latitude of its complement, 60°. Where the bearings are given in smaller fractions of degrees than is found in the table, the latitudes and departures can be found by interpolation.

Bearing Degrees	1		2		3		4		5	Bearing Degrees
	Lat.	Dep.	Lat.	Dep.	Lat.	Dep.	Lat.	Dep.	Lat.	
0	1.000	.000	2.000	.000	3.000	.000	4.000	.000	5.000	90
¼	1.000	.004	2.000	.009	3.000	.013	4.000	.017	5.000	89¾
½	1.000	.009	2.000	.017	3.000	.026	4.000	.035	5.000	89½
¾	1.000	.013	2.000	.026	3.000	.039	4.000	.052	5.000	89¼
1	1.000	.017	2.000	.035	3.000	.052	3.999	.070	4.999	89
1¼	1.000	.022	2.000	.044	2.999	.065	3.999	.087	4.999	88¾
1½	1.000	.026	1.999	.052	2.999	.079	3.999	.105	4.998	88½
1¾	1.000	.031	1.999	.061	2.999	.092	3.998	.122	4.998	88¼
2	.999	.035	1.999	.070	2.998	.105	3.998	.140	4.997	88
2¼	.999	.039	1.998	.079	2.998	.118	3.997	.157	4.996	87¾
2½	.999	.044	1.998	.087	2.997	.131	3.996	.174	4.995	87½
2¾	.999	.048	1.998	.096	2.997	.144	3.995	.192	4.994	87¼
3	.999	.052	1.997	.105	2.996	.157	3.995	.209	4.993	87
3¼	.998	.057	1.997	.113	2.995	.170	3.994	.227	4.992	86¾
3½	.998	.061	1.996	.122	2.994	.183	3.993	.244	4.991	86½
3¾	.998	.065	1.996	.131	2.994	.196	3.991	.262	4.989	86¼
4	.998	.070	1.995	.140	2.993	.209	3.990	.279	4.988	86
4¼	.997	.074	1.995	.148	2.992	.222	3.989	.296	4.986	85¾
4½	.997	.078	1.994	.157	2.991	.235	3.988	.314	4.985	85½
4¾	.997	.083	1.993	.166	2.990	.248	3.986	.331	4.983	85¼
5	.996	.087	1.992	.174	2.989	.261	3.985	.349	4.981	85
5¼	.996	.092	1.992	.183	2.987	.275	3.983	.366	4.979	84¾
5½	.995	.096	1.991	.192	2.986	.288	3.982	.383	4.977	84½
5¾	.995	.100	1.990	.200	2.985	.301	3.980	.401	4.975	84¼
6	.995	.105	1.989	.209	2.984	.314	3.978	.418	4.973	84
6¼	.994	.109	1.988	.218	2.982	.327	3.976	.435	4.970	83¾
6½	.994	.113	1.987	.226	2.981	.340	3.974	.453	4.968	83½
6¾	.993	.118	1.986	.235	2.979	.353	3.972	.470	4.965	83¼
7	.993	.122	1.985	.244	2.978	.366	3.970	.487	4.963	83
7¼	.992	.126	1.984	.252	2.976	.379	3.968	.505	4.960	82¾
7½	.991	.131	1.983	.261	2.974	.392	3.966	.522	4.957	82½
7¾	.991	.135	1.982	.270	2.973	.405	3.963	.539	4.954	82¼
8	.990	.139	1.981	.278	2.971	.418	3.961	.557	4.951	82
8¼	.990	.143	1.979	.287	2.969	.430	3.959	.574	4.948	81¾
8½	.989	.148	1.978	.296	2.967	.443	3.956	.591	4.945	81½
8¾	.988	.152	1.977	.304	2.965	.456	3.953	.608	4.942	81¼
9	.988	.156	1.975	.313	2.963	.469	3.951	.626	4.938	81
Bearing Degrees	Dep.	Lat.	Dep.	Lat.	Dep.	Lat.	Dep.	Lat.	Dep.	Bearing Degrees
	1		2		3		4		5	

Bearing Degrees	5	6		7		8		9		Bearing Degrees
	Dep.	Lat.	Dep.	Lat.	Dep.	Lat.	Dep.	Lat.	Dep.	
0	.000	6.000	.000	7.000	.000	8.000	.000	9.000	.000	90
¼	.022	6.000	.026	7.000	.031	8.000	.035	9.000	.039	89¾
½	.044	6.000	.052	7.000	.061	8.000	.070	9.000	.079	89½
¾	.065	5.999	.079	6.999	.092	7.999	.105	8.999	.118	89¼
1	.087	5.999	.105	6.999	.122	7.999	.140	8.999	.157	89
1¼	.109	5.999	.131	6.998	.153	7.998	.175	8.998	.196	88¾
1½	.131	5.998	.157	6.998	.183	7.997	.209	8.997	.236	88½
1¾	.153	5.997	.183	6.997	.214	7.996	.244	8.996	.275	88¼
2	.174	5.996	.209	6.996	.244	7.995	.279	8.995	.314	88
2¼	.196	5.995	.236	6.995	.275	7.994	.314	8.993	.353	87¾
2½	.218	5.994	.262	6.993	.305	7.992	.349	8.991	.393	87½
2¾	.240	5.993	.288	6.992	.336	7.991	.384	8.990	.432	87¼
3	.262	5.992	.314	6.990	.366	7.989	.419	8.988	.471	87
3¼	.283	5.990	.340	6.989	.397	7.987	.454	8.986	.510	86¾
3½	.305	5.989	.366	6.987	.427	7.985	.488	8.983	.549	86½
3¾	.327	5.987	.392	6.985	.458	7.983	.523	8.981	.589	86¼
4	.349	5.985	.419	6.983	.488	7.981	.558	8.978	.628	86
4¼	.371	5.984	.445	6.981	.519	7.978	.593	8.975	.667	85¾
4½	.392	5.982	.471	6.978	.549	7.975	.628	8.972	.706	85½
4¾	.414	5.979	.497	6.976	.580	7.973	.662	8.969	.745	85¼
5	.436	5.977	.523	6.973	.610	7.970	.697	8.966	.784	85
5¼	.458	5.975	.549	6.971	.641	7.966	.732	8.962	.824	84¾
5½	.479	5.972	.575	6.968	.671	7.963	.767	8.959	.863	84½
5¾	.501	5.970	.601	6.965	.701	7.960	.802	8.955	.902	84¼
6	.523	5.967	.627	6.962	.732	7.956	.836	8.951	.941	84
6¼	.544	5.964	.653	6.958	.762	7.952	.871	8.947	.980	83¾
6½	.566	5.961	.679	6.955	.792	7.949	.906	8.942	1.019	83½
6¾	.588	5.958	.705	6.951	.823	7.945	.940	8.938	1.058	83¼
7	.609	5.955	.731	6.948	.853	7:940	.975	8.933	1.097	83
7¼	.631	5.952	.757	6.944	.883	7.936	1.010	8.928	1.136	82¾
7½	.653	5.949	.783	6.940	.914	7.932	1.044	8.923	1.175	82½
7¾	.674	5.945	.809	6.936	.944	7.927	1.079	8.918	1.214	82¼
8	.696	5.942	.835	6.932	.974	7.922	1.113	8.912	1.253	82
8¼	.717	5.938	.861	6.928	1.004	7.917	1.148	8.907	1.291	81¾
8½	.739	5.934	.887	6.923	1.035	7.912	1.182	8.901	1.330	81½
8¾	.761	5.930	.913	6.919	1.065	7.907	1.217	8.895	1.369	81¼
9	.782	5.926	.939	6.914	1.095	7.902	1.251	8.889	1.408	81
	Lat.	Dep	Lat.	Dep	Lat.	Dep	Lat.	Dep	Lat.	
Bearing Degrees	5	6		7		8		9		Bearing Degrees

Bearing Degrees	1		2		3		4		5	Bearing Degrees
	Lat.	Dep.	Lat.	Dep.	Lat.	Dep.	Lat.	Dep.	Lat.	
9	.988	.156	1.975	.313	2.963	.469	3.951	.626	4.938	81
9¼	.987	.161	1.974	.321	2.961	.482	3.948	.643	4.935	80¾
9½	.986	.165	1.973	.330	2.959	.495	3.945	.660	4.931	80½
9¾	.986	.169	1.971	.339	2.957	.508	3.942	.677	4.928	80¼
10	.985	.174	1.970	.347	2.954	.521	3.939	.695	4.924	80
10¼	.984	.178	1.968	.356	2.952	.534	3.936	.712	4.920	79¾
10½	.983	.182	1.967	.364	2.950	.547	3.933	.729	4.916	79½
10¾	.982	.187	1.965	.373	2.947	.560	3.930	.746	4.912	79¼
11	.982	.191	1.963	.382	2.945	.572	3.927	.763	4.908	79
11¼	.981	.195	1.962	.390	2.942	.585	3.923	.780	4.904	78¾
11½	.980	.199	1.960	.399	2.940	.598	3.920	.797	4.900	78½
11¾	.979	.204	1.958	.407	2.937	.611	3.916	.815	4.895	78¼
12	.978	.208	1.956	.416	2.934	.624	3.913	.832	4.891	78
12¼	.977	.212	1.954	.424	2.932	.637	3.909	.849	4.886	77¾
12½	.976	.216	1.953	.433	2.929	.649	3.905	.866	4.881	77½
12¾	.975	.221	1.951	.441	2.926	.662	3.901	.883	4.877	77¼
13	.974	.225	1.949	.450	2.923	.675	3.897	.900	4.872	77
13¼	.973	.229	1.947	.458	2.920	.688	3.894	.917	4.867	76¾
13½	.972	.233	1.945	.467	2.917	.700	3.889	.934	4.862	76½
13¾	.971	.238	1.943	.475	2.914	.713	3.885	.951	4.857	76¼
14	.970	.242	1.941	.484	2.911	.726	3.881	.968	4.851	76
14¼	.969	.246	1.938	.492	2.908	.738	3.877	.985	4.846	75¾
14½	.968	.250	1.936	.501	2.904	.751	3.873	1.002	4.841	75½
14¾	.967	.255	1.934	.509	2.901	.764	3.868	1.018	4.835	75¼
15	.966	.259	1.932	.518	2.898	.776	3.864	1.035	4.830	75
15¼	.965	.263	1.930	.526	2.894	.789	3.859	1.052	4.824	74¾
15½	.964	.267	1.927	.534	2.891	.802	3.855	1.069	4.818	74½
15¾	.962	.271	1.925	.543	2.887	.814	3.850	1.086	4.812	74¼
16	.961	.276	1.923	.551	2.884	.827	3.845	1.103	4.806	74
16¼	.960	.280	1.920	.560	2.880	.839	3.840	1.119	4.800	73¾
16½	.959	.284	1.918	.568	2.876	.852	3.835	1.136	4.794	73½
16¾	.958	.288	1.915	.576	2.873	.865	3.830	1.153	4.788	73¼
17	.956	.292	1.913	.585	2.869	.877	3.825	1.169	4.782	73
17¼	.955	.297	1.910	.593	2.865	.890	3.820	1.186	4.775	72¾
17½	.954	.301	1.907	.601	2.861	.902	3.815	1.203	4.769	72½
17¾	.952	.305	1.905	.610	2.857	.915	3.810	1.220	4.762	72¼
18	.951	.309	1.902	.618	2.853	.927	3.804	1.236	4.755	72
Bearing Degrees	Dep.	Lat.	Dep.	Lat.	Dep.	Lat.	Dep.	Lat.	Dep.	Bearing Degrees
	1		2		3		4		5	

Bearing Degrees	5	6		7		8		9		Bearing Degrees
	Dep.	Lat.	Dep.	Lat.	Dep.	Lat.	Dep.	Lat.	Dep.	
9	.782	5.926	.939	6.914	1.095	7.902	1.251	8.889	1.408	81
9¼	.804	5.922	.964	6.909	1.125	7.896	1.286	8.883	1.447	80¾
9½	.825	5.918	.990	6.904	1.155	7.890	1.320	8.877	1.485	80½
9¾	.847	5.913	1.016	6.899	1.185	7.884	1.355	8.870	1.524	80¼
10	.868	5.909	1.042	6.894	1.216	7.878	1.389	8.863	1.563	80
10¼	.890	5.904	1.068	6.888	1.246	7.872	1.424	8.856	1.601	79¾
10½	.911	5.900	1.093	6.883	1.276	7.866	1.458	8.849	1.640	79½
10¾	.933	5.895	1.119	6.877	1.306	7.860	1.492	8.842	1.679	79¼
11	.954	5.890	1.145	6.871	1.336	7.853	1.526	8.835	1.717	79
11¼	.975	5.885	1.171	6.866	1.366	7.846	1.561	8.827	1.756	78¾
11½	.997	5.880	1.196	6.859	1.396	7.839	1.595	8.819	1.794	78½
11¾	1.018	5.874	1.222	6.853	1.425	7.832	1.629	8.811	1.833	78¼
12	1.040	5.869	1.247	6.847	1.455	7.825	1.663	8.803	1.871	78
12¼	1.061	5.863	1.273	6.841	1.485	7.818	1.697	8.795	1.910	77¾
12½	1.082	5.858	1.299	6.834	1.515	7.810	1.732	8.787	1.948	77½
12¾	1.103	5.852	1.324	6.827	1.545	7.803	1.766	8.778	1.986	77¼
13	1.125	5.846	1.350	6.821	1.575	7.795	1.800	8.769	2.025	77
13¼	1.146	5.840	1.375	6.814	1.604	7.787	1.834	8.760	2.063	76¾
13½	1.167	5.834	1.401	6.807	1.634	7.779	1.868	8.751	2.101	76½
13¾	1.188	5.828	1.426	6.799	1.664	7.771	1.902	8.742	2.139	76¼
14	1.210	5.822	1.452	6.792	1.693	7.762	1.935	8.733	2.177	76
14¼	1.231	5.815	1.477	6.785	1.723	7.754	1.969	8.723	2.215	75¾
14½	1.252	5.809	1.502	6.777	1.753	7.745	2.003	8.713	2.253	75½
14¾	1.273	5.802	1.528	6.769	1.782	7.736	2.037	8.703	2.291	75¼
15	1.294	5.796	1.553	6.761	1.812	7.727	2.071	8.693	2.329	75
15¼	1.315	5.789	1.578	6.754	1.841	7.718	2.104	8.683	2.367	74¾
15½	1.336	5.782	1.603	6.745	1.871	7.709	2.138	8.673	2.405	74½
15¾	1.357	5.775	1.629	6.737	1.900	7.700	2.172	8.662	2.443	74¼
16	1.378	5.768	1.654	6.729	1.929	7.690	2.205	8.651	2.481	74
16¼	1.399	5.760	1.679	6.720	1.959	7.680	2.239	8.640	2.518	73¾
16½	1.420	5.753	1.704	6.712	1.988	7.671	2.272	8.629	2.556	73½
16¾	1.441	5.745	1.729	6.703	2.017	7.661	2.306	8.618	2.594	73¼
17	1.462	5.738	1.754	6.694	2.047	7.650	2.339	8.607	2.631	73
17¼	1.483	5.730	1.779	6.685	2.076	7.640	2.372	8.595	2.669	72¾
17½	1.504	5.722	1.804	6.676	2.105	7.630	2.406	8.583	2.706	72½
17¾	1.524	5.714	1.829	6.667	2.134	7.619	2.439	8.572	2.744	72¼
18	1.545	5.706	1.854	6.657	2.163	7.608	2.472	8.560	2.781	72
	Lat.	Dep.	Lat.	Dep.	Lat.	Dep.	Lat.	Dep.	Lat.	
Bearing Degrees	5	6		7		8		9		Bearing Degrees

Bearing Degrees	1		2		3		4		5	Bearing Degrees
	Lat.	Dep.	Lat.	Dep.	Lat.	Dep.	Lat.	Dep.	Lat.	
18	.951	.309	1.902	.618	2.853	.927	3.804	1.236	4.755	72
18¼	.950	.313	1.899	.626	2.849	.939	3.799	1.253	4.748	71¾
18½	.948	.317	1.897	.635	2.845	.952	3.793	1.269	4.742	71½
18¾	.947	.321	1.894	.643	2.841	.964	3.788	1.286	4.735	71¼
19	.946	.326	1.891	.651	2.837	.977	3.782	1.302	4.728	71
19¼	.944	.330	1.888	.659	2.832	.989	3.776	1.319	4.720	70¾
19½	.943	.334	1.885	.668	2.828	1.001	3.771	1.335	4.713	70½
19¾	.941	.338	1.882	.676	2.824	1.014	3.765	1.352	4.706	70¼
20	.940	.342	1.879	.684	2.819	1.026	3.759	1.368	4.698	70
20¼	.938	.346	1.876	.692	2.815	1.038	3.753	1.384	4.691	69¾
20½	.937	.350	1.873	.700	2.810	1.051	3.747	1.401	4.683	69½
20¾	.935	.354	1.870	.709	2.805	1.063	3.741	1.417	4.676	69¼
21	.934	.358	1.867	.717	2.801	1.075	3.734	1.433	4.668	69
21¼	.932	.362	1.864	.725	2.796	1.087	3.728	1.450	4.660	68¾
21½	.930	.367	1.861	.733	2.791	1.100	3.722	1.466	4.652	68½
21¾	.929	.371	1.858	.741	2.786	1.112	3.715	1.482	4.644	68¼
22	.927	.375	1.854	.749	2.782	1.124	3.709	1.498	4.636	68
22¼	.926	.379	1.851	.757	2.777	1.136	3.702	1.515	4.628	67¾
22½	.924	.383	1.848	.765	2.772	1.148	3.696	1.531	4.619	67½
23¾	.922	.387	1.844	.773	2.767	1.160	3.689	1.547	4.611	67¼
23	.921	.391	1.841	.781	2.762	1.172	3.682	1.563	4.603	67
23¼	.919	.395	1.838	.789	2.756	1.184	3.675	1.579	4.594	66¾
23½	.917	.399	1.834	.797	2.751	1.196	3.668	1.595	4.585	66½
23¾	.915	.403	1.831	.805	2.746	1.208	3.661	1.611	4.577	66¼
24	.914	.407	1.827	.813	2.741	1.220	3.654	1.627	4.568	66
24¼	.912	.411	1.824	.821	2.735	1.232	3.647	1.643	4.559	65¾
24½	.910	.415	1.820	.829	2.730	1.244	3.640	1.659	4.550	65½
24¾	.908	.419	1.816	.837	2.724	1.256	3.633	1.675	4.541	65¼
25	.906	.423	1.813	.845	2.719	1.268	3.625	1.690	4.532	65
25¼	.904	.427	1.809	.853	2.713	1.280	3.618	1.706	4.522	64¾
25½	.903	.431	1.805	.861	2.708	1.292	3.610	1.722	4.513	64½
25¾	.901	.434	1.801	.869	2.702	1.303	3.603	1.738	4.503	64¼
26	.899	.438	1.798	.877	2.696	1.315	3.595	1.753	4.494	64
26¼	.897	.442	1.794	.885	2.691	1.327	3.587	1.769	4.484	63¾
26½	.895	.446	1.790	.892	2.685	1.339	3.580	1.785	4.475	63½
26¾	.893	.450	1.786	.900	2.679	1.350	3.572	1.800	4.465	63¼
27	.891	.454	1.782	.908	2.673	1.362	3.564	1.816	4.455	63
Bearing Degrees	Dep.	Lat.	Dep.	Lat.	Dep.	Lat.	Dep.	Lat.	Dep.	Bearing Degrees
	1		2		3		4		5	

Bearing Degrees	5 Dep.	6 Lat.	6 Dep.	7 Lat.	7 Dep.	8 Lat.	8 Dep.	9 Lat.	9 Dep.	Bearing Degrees
18	1.545	5.706	1.854	6.657	2.163	7.608	2.472	8.560	2.781	72
18¼	1.566	5.698	1.879	6.648	2.192	7.598	2.505	8.547	2.818	71¾
18½	1.587	5.690	1.904	6.638	2.221	7.587	2.538	8.535	2.856	71½
18¾	1.607	5.682	1.929	6.629	2.250	7.575	2.572	8.522	2.893	71¼
19	1.628	5.673	1.953	6.619	2.279	7.564	2.605	8.510	2.930	71
19¼	1.648	5.665	1.978	6.609	2.308	7.553	2.638	8.497	2.967	70¾
19½	1.669	5.656	2.003	6.598	2.337	7.541	2.670	8.484	3.004	70½
19¾	1.690	5.647	2.028	6.588	2.365	7.529	2.703	8.471	3.041	70¼
20	1.710	5.638	2.052	6.578	2.394	7.518	2.736	8.457	3.078	70
20¼	1.731	5.629	2.077	6.567	2.423	7.506	2.769	8.444	3.115	69¾
20½	1.751	5.620	2.101	6.557	2.451	7.493	2.802	8.430	3.152	69½
20¾	1.771	5.611	2.126	6.546	2.480	7.481	2.834	8.416	3.189	69¼
21	1.792	5.601	2.150	6.535	2.509	7.469	2.867	8.402	3.225	69
21¼	1.812	5.592	2.175	6.524	2.537	7.456	2.900	8.388	3.262	68¾
21½	1.833	5.582	2.199	6.513	2.566	7.443	2.932	8.374	3.299	68½
21¾	1.853	5.573	2.223	6.502	2.594	7.430	2.964	8.359	3.335	68¼
22	1.873	5.563	2.248	6.490	2.622	7.417	2.997	8.345	3.371	68
22¼	1.893	5.553	2.272	6.479	2.651	7.404	3.029	8.330	3.408	67¾
22½	1.913	5.543	2.296	6.467	2.679	7.391	3.061	8.315	3.444	67½
22¾	1.934	5.533	2.320	6.455	2.707	7.378	3.094	8.300	3.480	67¼
23	1.954	5.523	2.344	6.444	2.735	7.364	3.126	8.285	3.517	67
23¼	1.974	5.513	2.368	6.432	2.763	7.350	3.158	8.269	3.553	66¾
23½	1.994	5.502	2.392	6.419	2.791	7.336	3.190	8.254	3.589	66½
23¾	2.014	5.492	2.416	6.407	2.819	7.322	3.222	8.238	3.625	66¼
24	2.034	5.481	2.440	6.395	2.847	7.308	3.254	8.222	3.661	66
24¼	2.054	5.471	2.464	6.382	2.875	7.294	3.286	8.206	3.696	65¾
24½	2.073	5.460	2.488	6.370	2.903	7.280	3.318	8.190	3.732	65½
24¾	2.093	5.449	2.512	6.357	2.931	7.265	3.349	8.173	3.768	65¼
25	2.113	5.438	2.536	6.344	2.958	7.250	3.381	8.157	3.804	65
25¼	2.133	5.427	2.559	6.331	2.986	7.236	3.413	8.140	3.839	64¾
25½	2.153	5.416	2.583	6.318	3.014	7.221	3.444	8.123	3.875	64½
25¾	2.172	5.404	2.607	6.305	3.041	7.206	3.476	8.106	3.910	64¼
26	2.192	5.393	2.630	6.292	3.069	7.190	3.507	8.089	3.945	64
26¼	2.211	5.381	2.654	6.278	3.096	7.175	3.538	8.072	3.981	63¾
26½	2.231	5.370	2.677	6.265	3.123	7.160	3.570	8.054	4.016	63½
26¾	2.250	5.358	2.701	6.251	3.151	7.144	3.601	8.037	4.051	63¼
27	2.270	5.346	2.724	6.237	3.178	7.128	3.632	8.019	4.086	63
Bearing Degrees	5 Lat.	6 Dep.	6 Lat.	7 Dep.	7 Lat.	8 Dep.	8 Lat.	9 Dep.	9 Lat.	Bearing Degrees

Bearing Degrees	1		2		3		4		5	Bearing Degrees
	Lat.	Dep.	Lat.	Dep.	Lat.	Dep.	Lat.	Dep.	Lat.	
27	.891	.454	1.782	.908	2.673	1.362	3.564	1.816	4.455	63
27¼	.889	.458	1.778	.916	2.667	1.374	3.556	1.831	4.445	62¾
27½	.887	.462	1.774	.923	2.661	1.385	3.548	1.847	4.435	62½
27¾	.885	.466	1.770	.931	2.655	1.397	3.540	1.862	4.425	62¼
28	.883	.469	1.766	.939	2.649	1.408	3.532	1.878	4.415	62
28¼	.881	.473	1.762	.947	2.643	1.420	3.524	1.893	4.404	61¾
28½	.879	.477	1.758	.954	2.636	1.431	3.515	1.909	4.394	61½
28¾	.877	.481	1.753	.962	2.630	1.443	3.507	1.924	4.384	61¼
29	.875	.485	1.749	.970	2.624	1.454	3.498	1.939	4.373	61
29¼	.872	.489	1.745	.977	2.617	1.466	3.490	1.954	4.362	60¾
29½	.870	.492	1.741	.985	2.611	1.477	3.481	1.970	4.352	60½
29¾	.868	.496	1.736	.992	2.605	1.489	3.473	1.985	4.341	60¼
30	.866	.500	1.732	1.000	2.598	1.500	3.464	2.000	4.330	60
30¼	.864	.504	1.728	1.008	2.592	1.511	3.455	2.015	4.319	59¾
30½	.862	.508	1.723	1.015	2.585	1.523	3.447	2.030	4.308	59½
20¾	.859	.511	1.719	1.023	2.578	1.534	3.438	2.045	4.297	59¼
31	.857	.515	1.714	1.030	2.572	1.545	3.429	2.060	4.286	59
31¼	.855	.519	1.710	1.038	2.565	1.556	3.420	2.075	4.275	58¾
31½	.853	.522	1.705	1.045	2.558	1.567	3.411	2.090	4.263	58½
31¾	.850	.526	1.701	1.052	2.551	1.579	3.401	2.105	4.252	58¼
32	.848	.530	1.696	1.060	2.544	1.590	3.392	2.120	4.240	58
32¼	.846	.534	1.691	1.067	2.537	1.601	3.383	2.134	4.229	57¾
32½	.843	.537	1.687	1.075	2.530	1.612	3.374	2.149	4.217	57½
32¾	.841	.541	1.682	1.082	2.523	1.623	3.364	2.164	4.205	57¼
33	.839	.545	1.677	1.089	2.516	1.634	3.355	2.179	4.193	57
33¼	.836	.548	1.673	1.097	2.509	1.645	3.345	2.193	4.181	56¾
33½	.834	.552	1.668	1.104	2.502	1.656	3.336	2.208	4.169	56½
33¾	.831	.556	1.663	1.111	2.494	1.667	3.326	2.222	4.157	56¼
34	.829	.559	1.658	1.118	2.487	1.678	3.316	2.237	4.145	56
34¼	.827	.563	1.653	1.126	2.480	1.688	3.306	2.251	4.133	55¾
34½	.824	.566	1.648	1.133	2.472	1.699	3.297	2.266	4.121	55½
34¾	.822	.570	1.643	1.140	2.465	1.710	3.287	2.280	4.108	55¼
35	.819	.574	1.638	1.147	2.457	1.721	3.277	2.294	4.096	55
35¼	.817	.577	1.633	1.154	2.450	1.731	3.267	2.309	4.083	54¾
35½	.814	.581	1.628	1.161	2.442	1.742	3.257	2.323	4.071	54½
35¾	.812	.584	1.623	1.168	2.435	1.753	3.246	2.337	4.058	54¼
36	.809	.588	1.618	1.176	2.427	1.763	3.236	2.351	4.045	54
Bearing Degrees	Dep	Lat.	Dep	Lat.	Dep	Lat.	Dep	Lat.	Dep	Bearing Degrees
	1		2		3		4		5	

Bearing Degrees	5	6		7		8		9		Bearing Degrees
	Dep.	Lat.	Dep.	Lat.	Dep.	Lat.	Dep.	Lat.	Dep.	
27	2.270	5.346	2.724	6.237	3.178	7.128	3.632	8.019	4.086	63
27¼	2.289	5.334	2.747	6.223	3.205	7.112	3.663	8.001	4.121	62¾
27½	2.309	5.322	2.770	6.209	3.232	7.096	3.694	7.983	4.156	62½
27¾	2.328	5.310	2.794	6.195	3.259	7.080	3.725	7.965	4.190	62¼
28	2.347	5.298	2.817	6.181	3.286	7.064	3.756	7.947	4.225	62
28¼	2.367	5.285	2.840	6.166	3.313	7.047	3.787	7.928	4.260	61¾
28½	2.386	5.273	2.863	6.152	3.340	7.031	3.817	7.909	4.294	61½
28¾	2.405	5.260	2.886	6.137	3.367	7.014	3.848	7.891	4.329	61¼
29	2.424	5.248	2.909	6.122	3.394	6.997	3.878	7.872	4.363	61
29¼	2.443	5.235	2.932	6.107	3.420	6.980	3.909	7.852	4.398	60¾
29½	2.462	5.222	2.955	6.093	3.447	6.963	3.939	7.833	4.432	60½
29¾	2.481	5.209	2.977	6.077	3.474	6.946	3.970	7.814	4.466	60¼
30	2.500	5.196	3.000	6.062	3.500	6.928	4.000	7.794	4.500	60
30¼	2.519	5.183	3.023	6.047	3.526	6.911	4.030	7.775	4.534	59¾
30½	2.538	5.170	3.045	6.031	3.553	6.893	4.060	7.755	4.568	59½
30¾	2.556	5.156	3.068	6.016	3.579	6.875	4.090	7.735	4.602	59¼
31	2.575	5.143	3.090	6.000	3.605	6.857	4.120	7.715	4.635	59
31¼	2.594	5.129	3.113	5.984	3.631	6.839	4.150	7.694	4.669	58¾
31½	2.612	5.116	3.135	5.968	3.657	6.821	4.180	7.674	4.702	58½
31¾	2.631	5.102	3.157	5.952	3.683	6.803	4.210	7.653	4.736	58¼
32	2.650	5.088	3.180	5.936	3.709	6.784	4.239	7.632	4.769	58
32¼	2.668	5.074	3.202	5.920	3.735	6.766	4.269	7.612	4.802	57¾
32½	2.686	5.060	3.224	5.904	3.761	6.747	4.298	7.591	4.836	57½
32¾	2.705	5.046	3.246	5.887	3.787	6.728	4.328	7.569	4.869	57¼
33	2.723	5.032	3.268	5.871	3.812	6.709	4.357	7.548	4.902	57
33¼	2.741	5.018	3.290	5.854	3.838	6.690	4.386	7.527	4.935	56¾
33½	2.760	5.003	3.312	5.837	3.864	6.671	4.416	7.505	4.967	56½
33¾	2.778	4.989	3.333	5.820	3.889	6.652	4.445	7.483	5.000	56¼
34	2.796	4.974	3.355	5.803	3.914	6.632	4.474	7.461	5.033	56
34¼	2.814	4.960	3.377	5.786	3.940	6.613	4.502	7.439	5.065	55¾
34½	2.832	4.945	3.398	5.769	3.965	6.593	4.531	7.417	5.098	55½
34¾	2.850	4.930	3.420	5.752	3.990	6.573	4.560	7.395	5.130	55¼
35	2.868	4.915	3.441	5.734	4.015	6.553	4.589	7.372	5.162	55
35¼	2.886	4.900	3.463	5.716	4.040	6.533	4.617	7.350	5.194	54¾
35½	2.904	4.885	3.484	5.699	4.065	6.513	4.646	7.327	5.226	54½
35¾	2.921	4.869	3.505	5.681	4.090	6.493	4.674	7.304	5.258	54¼
36	2.939	4.854	3.527	5.663	4.115	6.472	4.702	7.281	5.290	54
	Lat.	Dep.	Lat.	Dep.	Lat.	Dep.	Lat.	Dep.	Lat.	
Bearing Degrees	5	6		7		8		9		Bearing Degrees

Bearing Degrees	1		2		3		4		5	Bearing Degrees
	Lat.	Dep.	Lat.	Dep.	Lat.	Dep.	Lat.	Dep.	Lat.	
36	.809	.588	1.618	1.176	2.427	1.763	3.236	2.351	4.045	54
36¼	.806	.591	1.613	1.183	2.419	1.774	3.226	2.365	4.032	53¾
36½	.804	.595	1.608	1.190	2.412	1.784	3.215	2.379	4.019	53½
36¾	.801	.598	1.603	1.197	2.404	1.795	3.205	2.393	4.006	53¼
37	.799	.602	1.597	1.204	2.396	1.805	3.195	2.407	3.993	53
37¼	.796	.605	1.592	1.211	2.388	1.816	3.184	2.421	3.980	52¾
37½	.793	.609	1.587	1.218	2.380	1.826	3.173	2.435	3.967	52½
37¾	.791	.612	1.581	1.224	2.372	1.837	3.163	2.449	3.953	52¼
38	.788	.616	1.576	1.231	2.364	1.847	3.152	2.463	3.940	52
38¼	.785	.619	1.571	1.238	2.356	1.857	3.141	2.476	3.927	51¾
38½	.783	.623	1.565	1.245	2.348	1.868	3.130	2.490	3.913	51½
38¾	.780	.626	1.560	1.252	2.340	1.878	3.120	2.504	3.899	51¼
39	.777	.629	1.554	1.259	2.331	1.888	3.109	2.517	3.886	51
39¼	.774	.633	1.549	1.265	2.323	1.898	3.098	2.531	3.872	50¾
39½	.772	.636	1.543	1.272	2.315	1.908	3.086	2.544	3.858	50½
39¾	.769	.639	1.538	1.279	2.307	1.918	3.075	2.558	3.844	50¼
40	.766	.643	1.532	1.286	2.298	1.928	3.064	2.571	3.830	50
40¼	.763	.646	1.526	1.292	2.290	1.938	3.053	2.584	3.816	49¾
40½	.760	.649	1.521	1.299	2.281	1.948	3.042	2.598	3.802	49½
40¾	.758	.653	1.515	1.306	2.273	1.958	3.030	2.611	3.788	49¼
41	.755	.656	1.509	1.312	2.264	1.968	3.019	2.624	3.774	49
41¼	.752	.659	1.504	1.319	2.256	1.978	3.007	2.637	3.759	48¾
41½	.749	.663	1.498	1.325	2.247	1.988	2.996	2.650	3.745	48½
41¾	.746	.666	1.492	1.332	2.238	1.998	2.984	2.664	3.730	48¼
42	.743	.669	1.486	1.338	2.229	2.007	2.973	2.677	3.716	48
42¼	.740	.672	1.480	1.345	2.221	2.017	2.961	2.689	3.701	47¾
42½	.737	.676	1.475	1.351	2.212	2.027	2,949	2.702	3.686	47½
42¾	.734	.679	1.469	1.358	2.203	2.036	2.937	2.715	3.672	47¼
43	.731	.682	1.463	1.364	2.194	2.046	2.925	2.728	3.657	47
43¼	.728	.685	1.457	1.370	2.185	2.056	2.913	2.741	3.642	46¾
43½	.725	.688	1.451	1.377	2.176	2.065	2.901	2.753	3.627	46½
43¾	.722	.692	1.445	1.383	2.167	2.075	2.889	2.766	3.612	46¼
44	.719	.695	1.439	1.389	2.158	2.084	2.877	2.779	3.597	46
44¼	.716	.698	1.433	1.396	2.149	2.093	2.865	2.791	3.582	45¾
44½	.713	.701	1.427	1.402	2.140	2.103	2.853	2.804	3.566	45½
44¾	.710	.704	1.420	1.408	2.131	2.112	2.841	2.816	3.551	45¼
45	.707	.707	1.414	1.414	2.121	2.121	2.828	2.828	3.536	45
Bearing Degrees	Dep.	Lat.	Dep.	Lat.	Dep.	Lat.	Dep.	Lat.	Dep.	Bearing Degrees
	1		2		3		4		5	

Bearing Degrees	5	6		7		8		9		Bearing Degrees
	Dep.	Lat.	Dep.	Lat.	Dep.	Lat.	Dep.	Lat.	Dep.	
36	2.939	4.854	3.527	5.663	4.115	6.472	4.702	7.281	5.290	54
36¼	2.957	4.839	3.548	5.645	4.139	6.452	4.730	7.258	5.322	53¾
36½	2.974	4.823	3.569	5.627	4.164	6.431	4.759	7.235	5.353	53½
36¾	2.992	4.808	3.590	5.609	4.188	6.410	4.787	7.211	5.385	53¼
37	3.009	4.792	3.611	5.590	4.213	6.389	4.815	7.188	5.416	53
37¼	3.026	4.776	3.632	5.572	4.237	6.368	4.842	7.164	5.448	52¾
37½	3.044	4.760	3.653	5.554	4.261	6.347	4.870	7.140	5.479	52½
37¾	3.061	4.744	3 673	5.535	4.286	6.326	4.898	7.116	5.510	52¼
38	3.078	4.728	3.694	5.516	4.310	6.304	4.925	7.092	5.541	52
38¼	3.095	4.712	3.715	5.497	4.334	6.283	4.953	7.068	5.572	51¾
38½	3.113	4.696	3.735	5.478	4.358	6.261	4.980	7.043	5.603	51½
38¾	3.130	4.679	3.756	5.459	4.381	6.239	5.007	7.019	5.663	51¼
39	3.147	4.663	3.776	5.440	4.405	6.217	5.035	6.994	5.664	51
39¼	3.164	4.646	3.796	5.421	4.429	6.195	5.062	6.970	5.694	50¾
39½	3.180	4.630	3.816	5.401	4.453	6.173	5.089	6.945	5.725	50½
39¾	3.197	4.613	3.837	5.382	4.476	6.151	5.116	6.920	5.755	50¼
40	3.214	4.596	3.857	5.362	4.500	6.128	5.142	6.894	5.785	50
40¼	3.231	4.579	3.877	5.343	4.523	6.106	5.169	6.869	5.815	49¾
40½	3.247	4.562	3.897	5.323	4.546	6.083	5.196	6.844	5.845	49½
40¾	3.264	4.545	3.917	5.303	4.569	6.061	5.222	6.818	5.875	49¼
41	3.280	4.528	3.936	5.283	4.592	6.038	5.248	6.792	5.905	49
41¼	3.297	4.511	3.956	5.263	4.615	6.015	5.275	6.767	5.934	48¾
41½	3.313	4.494	3.976	5.243	4.638	5.992	5.301	6.741	5.964	48½
41¾	3.329	4.476	3.995	5.222	4.661	5.968	5.327	6.715	5.993	48¼
42	3.346	4.459	4.015	5.202	4.684	5.945	5.353	6.688	6.022	48
42¼	3.362	4.441	4.034	5.182	4.707	5.922	5.379	6.662	6.051	47¾
42½	3.378	4.424	4.054	5.161	4.729	5.898	5.405	6.635	6.080	47½
42¾	3.394	4.406	4.073	5.140	4.752	5.875	5.430	6.609	6.109	47¼
43	3.410	4.388	4.092	5.119	4.774	5.851	5.456	6.582	6.138	47
43¼	3.426	4.370	4.111	5.099	4.796	5.827	5.481	6.555	6.167	46¾
43½	3.442	4.352	4.130	5.078	4.818	5.803	5.507	6.528	6.195	46½
43¾	3 458	4.334	4.149	5.057	4.841	5.779	5.532	6.501	6.224	46¼
44	3.473	4.316	4.168	5.035	4.863	5.755	5.557	6.474	6.252	46
44¼	3.489	4.298	4.187	5.014	4.885	5.730	5.582	6.447	6.280	45¾
44½	3.505	4.280	4.206	4.993	4.906	5.706	5.607	6.419	6.308	45½
44¾	3.520	4.261	4.224	4.971	4.928	5.681	5.632	6.392	6.336	45¼
45	3.536	4.243	4.243	4.950	4.950	5.657	5.657	6.364	6.364	45
	Lat.	Dep.	Lat.	Dep.	Lat.	Dep.	Lat.	Dep.	Lat.	
Bearing Degrees	5	6		7		8		9		Bearing Degrees

MATHEMATICS

SIMPLE PROPORTION, OR SINGLE RULE OF THREE

A proportion is an expression of equality between equal ratios; thus, the ratio of 10 to 5 = the ratio of 4 to 2, and is expressed thus: $10:5=4:2$. There are four terms in proportion; the first and last are the *extremes* and the second and third are the *means*.

Quantities are in proportion by *alternation* when antecedent is compared with antecedent and consequent with consequent; thus, if $10:5=4:2$, then $10:4=5:2$. Quantities are in proportion by *inversion* when the antecedents are made consequents and the consequents antecedents; thus, if $10:5=4:2$, then $5:10=2:4$. In any proportion, the product of the means will equal the product of the extremes, thus, if $10:5=4:2$, then $5\times4=10\times2$.

A mean proportional between two quantities equals the square root of their product; thus, a mean proportional between 12 and 3 = the square root of 12×3, or 6.

If the two means and one extreme of a proportion are given, the other extreme may be found by dividing the product of the means by the given extreme. Thus, $10:5=4:(\ )$, then $(4\times5)\div10=2$, and the proportion is $10:5=4:2$. If the two extremes and one mean are given, the other mean may be found by dividing the product of the extremes by the given mean. Thus, $10:(\ )=4:2$, then $(10\times2)\div4=5$, and the proportion is $10:5=4:2$.

Example.—If 6 men load 30 wagons of coal in a day, how many wagons will 10 men load?

Solution.—They will evidently load more, so the second term of the proportion must be greater than the first.

$6:10=30:(\ )$; then, $(10\times30)\div6=50$.

PERCENTAGE

Percentage means by or on the hundred. Thus, $1\% = \frac{1}{100} = .01$, $3\% = \frac{3}{100} = .03$.

To Find the Percentage, Having the Rate and the Base. Multiply the base by the rate expressed in hundredths; thus, 6% of 1,930 is $1{,}930 \times .06 = 115.80$.

To Find the Amount, Having the Base and the Rate.—Multiply the base by 1 plus the rate; thus, the amount of \$1,930 for 1 yr. at 6% is $\$1{,}930 \times 1.06 = \$2{,}045.80$.

To Find the Base, Having the Rate and the Percentage. Divide the percentage by the rate; thus, if the rate is 6% and the percentage is 115.80, the base is $115.80 \div .06 = 1{,}930$.

To Find the Rate, Having the Percentage and the Base. Divide the percentage by the base; thus, if the percentage is 115.80 and the base 1,930, the rate is $115.80 \div 1{,}930 = .06$, or 6%.

FORMULAS

The term *formula*, as used in mathematics and in technical books, may be defined as a rule in which symbols are used instead of words; in fact, a formula may be regarded as a shorthand method of expressing a rule. The signs used are the ordinary signs indicative of operations and the signs of aggregation; all of which are used in arithmetic.

The use of formulas can best be shown by means of an example; therefore, the well-known rule for finding the horsepower of a steam engine will be taken. This rule may be stated as follows:

Rule.—*Divide the continued product of the mean effective pressure, in pounds per square inch, the length of the stroke, in feet, the area of the piston, in square inches, and the number of strokes per minute by 33,000; the result will be the horsepower.*

An examination of the rule will show that four quantities (viz., the mean effective pressure, the length of the stroke, the area of the piston, and the number of strokes) are multiplied together, and the result is divided by 33,000. Hence, the rule might be expressed as follows:

$$\text{Horsepower} = \frac{\text{mean effective pressure}}{\text{(in pounds per square inch)}} \times \frac{\text{stroke}}{\text{(in feet)}} \times \frac{\text{area of piston}}{\text{(in square inches)}} \times \frac{\text{number of strokes}}{\text{(per minute)}} \div 33{,}000.$$

This expression can be greatly shortened by representing each quantity by a single letter, thus representing horsepower by the letter H, the mean effective pressure, in pounds per square inch, by P, the length of the stroke in feet, by L, the area of the piston, in square inches, by A, the number of strokes per minute by N, and substituting these letters for the quantities that they represent, the following *formula* is obtained,

$$H = \frac{P \times L \times A \times N}{33{,}000}$$

The formula just given shows that a formula is really a shorthand method of expressing a rule. It is customary, however, to omit the sign of multiplication between two or more quantities when they are to be multiplied together, or between a number and a letter representing a quantity, it being always understood that when two letters are adjacent with no sign between them, the quantities represented by these letters are to be multiplied. Bearing this fact in mind, the formula just given can be further simplified to

$$H = \frac{PLAN}{33{,}000}$$

The sign of multiplication, evidently, cannot be omitted between two or more numbers, as it would then be impossible to distinguish the numbers.

Use of Formulas.—The area of any segment of a circle that is less than (or equal to) a semicircle is expressed by the formula

$$A = \frac{\pi r^2 E}{360} - \frac{c}{2}(r - h),$$

in which A = area of segment;

$\pi = 3.1416$;

r = radius;

E = angle obtained by drawing lines from center to extremities of arc of segment;

c = chord of segment;

h = height of segment.

EXAMPLE.—What is the area of a segment whose chord is 10 in. long, angle subtended by chord is 83.46°, radius, is 7.5 in., and height of segment is 1.91 in.?

SOLUTION.—Applying the formula just given,

$$A = \frac{3.1416 \times 7.5^2 \times 83.46}{360} - \frac{10}{2} \times (7.5 - 1.91)$$

$$= 40.968 - 27.95 = 13.018 \text{ sq. in., nearly}$$

The area of any triangle may be found by means of the following formula,

$$A = \frac{b}{2}\sqrt{a^2 - \left(\frac{a^2 + b^2 - c^2}{2b}\right)^2},$$

in which A = area;

a, b, and c = lengths of sides.

EXAMPLE.—What is the area of a triangle whose sides are 21 ft., 46 ft., and 50 ft. long?

SOLUTION.—In order to apply the formula, let a represent the side that is 21 ft. long; b, the side that is 50 ft. long; and c, the side that is 46 ft. long. Then, substituting,

$$A = \frac{50}{2}\sqrt{21^2 - \left(\frac{21^2 + 50^2 - 46^2}{2 \times 50}\right)^2}$$

$$= \frac{50}{2}\sqrt{441 - \left(\frac{441 + 2{,}500 - 2{,}116}{100}\right)^2} = 25\sqrt{441 - \left(\frac{825}{100}\right)^2}$$

$$= 25\sqrt{441 - 8.25^2} = 25.\sqrt{441 - 68.0625} = 25\sqrt{372.9375}$$

$$= 25 \times 19.312 = 482.8 \text{ sq. ft., nearly}$$

These operations have been extended much further than was necessary; this was done in order to show the reader every step of the process.

Rankine-Gordon Formula.—The Rankine-Gordon formula for determining the least load in pounds that will cause a long column to break is

$$P = \frac{SA}{1 + q\frac{l^2}{G^2}},$$

in which P = load (pressure), in pounds;

S = ultimate strength of material composing column, in pounds per square inch;

A = area of cross-section of column, in square inches;
q = a factor (multiplier) whose value depends on shape of ends of column and on material composing column;
l = length of column, in inches;
G = least radius of gyration of cross-section of column.

EXAMPLE.—What is the least load that will break a hollow steel column whose outside diameter is 14 in., inside diameter 11 in., length 20 ft., and whose ends are flat?

SOLUTION.—For steel, $S=150{,}000$, and $q=\frac{1}{25{,}000}$ for flat-ended steel columns; $A=.7854(d_1^2-d_2^2)$, d_1 and d_2 being the outside and inside diameters, respectively; $l=20\times12=240$ in.; and $G^2=\frac{d_1^2+d_2^2}{16}$. Substituting these values in the formula,

$$P=\frac{150{,}000\times.7854(14^2-11^2)}{1+\frac{1}{25{,}000}\times\frac{240^2}{\frac{14^2+11^2}{16}}}=\frac{150{,}000\times58.905}{1+.1163}$$

$$=\frac{8{,}835.750}{1.1163}=7{,}915{,}211 \text{ lb.}$$

LOGARITHMS

Logarithms are designed to diminish the labor of multiplication and division, by substituting in their stead addition and subtraction. A *logarithm* is the exponent of the power to which a fixed number, called the *base*, must be raised to produce a given number. The base of the common system is 10, and, as a logarithm is the exponent of the power to which the base must be raised in order to be equal to a given number, all numbers are to be regarded as powers of 10; hence,

$10^0=1$, therefore logarithm of $1=0$
$10^1=10$, therefore logarithm of $10=1$
$10^2=100$, therefore logarithm of $100=2$
$10^3=1{,}000$, therefore logarithm of $1{,}000=3$
$10^4=10{,}000$, therefore logarithm of $10{,}000=4$

The logarithms of numbers between 1 and 10 are less than unity, and are expressed as decimals. The logarithm of any number between 10 and 100 is more than 1 and less than 2, hence it is equal to 1 plus a decimal. Between 100 and 1,000 it is equal to 2 plus a decimal, etc.

The integral part of a logarithm is its *characteristic*, the decimal part is its *mantissa*. For example, the log of 67.7 is 1.83059; the characteristic of this logarithm is 1 and the mantissa is .83059. The characteristic of a logarithm is always 1 less than the number of whole figures expressing that number, and may be either negative or positive. The characteristic of the logarithm of 7 is 0; of 17 is 1; of 717 is 2; etc. The mantissa is always considered positive.

To Find Logarithm of Any Number Between 1 and 100. Look on the first page of the table, along the column marked No., for the given number; opposite it will be found the logarithm with its charactcristic.

To Find Logarithm of Any Number of Three Figures.—Find the decimal in the first column to the right of the number; prefix to this the characteristic 2. Thus, the logarithm of 327 is 2.51455. As the first two figures of the decimal are the same for several successive figures, they are only given where they change. Thus, the decimal part of the logarithm of 302 is .48001. The first two figures remain the same up to 310, and are therefore to be supplied.

To Find Logarithm of Any Number of Four Figures.—Look in the column headed No. for the first three figures, and then along the top of the page for the fourth figure. Down the column headed by the fourth figure, and opposite the first three, will be found the decimal part. To this prefix the characteristic 3.

To Find Logarithm of Any Number of More Than Four Figures.—Place a decimal point after the fourth figure from the left, thus changing the number into an integer and a decimal. If the decimal part contains more than two figures, and its second figure is 5 or greater, add 1 to the first figure in the decimal. Find the mantissa of the first four figures, and subtract it from the next greater mantissa in the table. Under the heading P. P., find a column headed by the difference first

found. Find in this column the number opposite the number corresponding to the first figure of the decimal, or the first figure increased by one, and add it to the mantissa already found for the first four figures of the given number.

EXAMPLE.—What is the logarithm of 234,567?

SOLUTION.—Placing a decimal point after the fourth figure from the left gives 2,345.67. The mantissa of 2,345 is .37014; the difference between .37014 and the next higher logarithm .37033 is 19. Add 1 to the first figure of the decimal 6, and in the column headed 19, under P. P., opposite 7, is found 13.3, which, added to the portion of the mantissa already found, .37014, gives .37027. The characteristic is 5, hence the logarithm is 5.37027.

To Find Logarithm of Decimal Fraction.—Proceed according to the rules just given, except in regard to the characteristic. Where the number consists of a whole number and a decimal, the characteristic is 1 less than the whole number. Where it is a simple decimal, or when there are no ciphers between the decimal point and the first numerator, the characteristic is negative, and is expressed by 1, with a minus sign over it. Where there is one cipher between the decimal point and first numerator, the characteristic is 2, with a minus sign over it. Where there are 2 ciphers, the characteristic is 3, with a minus sign over it. Thus:

The logarithm of 67.7 $= 1.83059$
The logarithm of 6.77 $= 0.83059$
The logarithm of .677 $= \bar{1}.83059$
The logarithm of .0677 $= \bar{2}.83059$
The logarithm of .00677 $= \bar{3}.83059$

The characteristic only is negative; the decimal part is positive.

To Find Logarithm of Vulgar Fraction.—Subtract the logarithm of the denominator from the logarithm of the numerator; the difference is the logarithm of the fraction.

EXAMPLE.—Find logarithm of $\frac{4}{10}$.

SOLUTION.—

$$\begin{array}{r} \text{Log } 4 = 0.60206 \\ \text{Log } 10 = 1. \\ \hline \bar{1}.60206 \end{array}$$

$\bar{1}.60206$ is the logarithm of .4.

To Find Natural Number Corresponding to Any Logarithm. Look in the column headed 0 for the first two figures of the decimal part; the other four figures are to be looked for in the same or in one of the nine following columns. If they are exactly found, the number must be made to correspond with the characteristic by pointing off decimals or annexing ciphers.

If the decimal portion cannot be found exactly, find the next lower logarithm, subtract it from the given logarithm, divide the difference by the difference between the next lower and the next higher logarithm, and annex the quotient to the natural number found opposite the lower logarithm.

To Multiply by Logarithms.—Add the logarithms of the factors together; the sum will be the logarithm of their product.

EXAMPLE.—$67.7 \times .677 = ?$

SOLUTION.—

$$\begin{array}{rl} \text{Log } 67.7 = & 1.83059 \\ \text{Log } .677 = & \underline{\bar{1}.83059} \\ & 1.66118 \end{array}$$

1.66118 is the logarithm of 45.833

To Divide by Logarithms.—Subtract the logarithm of the divisor from the logarithm of the dividend; the difference will be the logarithm of the quotient.

EXAMPLE.—Divide 67.7 by .0677.

SOLUTION.—

$$\begin{array}{rl} \text{Log } 67.7 = & 1.83059 \\ \text{Log } .0677 = & \underline{\bar{2}.83059} \\ & 3.00000 \end{array}$$

3 is the logarithm of 1,000

To Square a Number by Logarithms.—Multiply the logarithm of the number by 2; the product will be the logarithm of the square of the number.

EXAMPLE.—Square .677.

SOLUTION.—

$$\begin{array}{rl} \text{Log } .677 = & \bar{1}.83059 \\ & \underline{2} \\ & \bar{1}.66118 \end{array}$$

$\bar{1}.66118$ is the logarithm of .45833

To Cube a Number by Logarithms.—Multiply the logarithm of the number by 3; the product will be the logarithm of the cube of the number.

To Raise a Number to Any Power by Logarithms.—Multiply the logarithm of the number by 4, 5, 6, or 7, and the results will be the logarithms of the 4th, 5th, 6th, or 7th powers, respectively; thus, a number can readily be raised to any power required.

To Extract Any Root of a Number by Logarithms.—Divide the logarithm of the number by the index of the root required; the quotient will be the logarithm of the required root.

EXAMPLE.—Find the square root of 625.

SOLUTION.— Logarithm of $625 = 2.79588$
$2.79588 \div 2 = 1.39794$
$1.39794 =$ logarithm of 25

Therefore, the square root of 625 is 25.

To Divide a Logarithm Having a Negative Characteristic. If the characteristic is evenly divisible by the divisor, divide in the usual manner and retain the negative sign of the characteristic in the quotient. If the negative characteristic is less than, or is not evenly divisible by, the divisor, add such a negative number to it as will make it evenly divisible, and prefix an equal positive number to the decimal part of the logarithm; then divide the increased negative characteristic by the divisor, to obtain the characteristic of the quotient desired. To obtain the decimal part of the quotient, divide the decimal part of the logarithm, with the positive number prefixed, in the usual manner. To this quotient prefix the negative characteristic already found, and this will be the quotient desired. Logarithms are particularly useful in those cases where the unknown quantity is an exponent, or when the exponent is a decimal.

EXAMPLE 1.—Divide $\bar{6}.3246846$ by 3.

SOLUTION.— $\bar{6}.3246846 \div 3 = \bar{2}.1582282$

EXAMPLE 2.—Divide $\overline{14}.3268472$ by 9.

SOLUTION.— $14.3268472 \div 9 = (\overline{14} + \bar{4} = \overline{18}) + (4 + .3268472)$
$\times 18 + 4.3268472 \div 9 = \sqrt[5]{2.4807608}$

EXAMPLE 3.—Find $\sqrt[5]{.677}$.

SOLUTION.—

$$\sqrt[5]{.677} = \log .677 \div 5 = \bar{1}.830578 \div 5;\ \bar{5} \div 5 + 4.830589 \div 5$$
$$= \bar{1}.9661178 = .9249+$$

TABLE
OF
COMMON LOGARITHMS
OF NUMBERS
FROM 1 TO 10,000

No.	Log	No.	Log	No.	Log	No.	Log	No.	Log
0	– ∞	20	30 103	40	60 206	60	77 815	80	90 309
1	00 000	21	32 222	41	61 278	61	78 533	81	90 849
2	30 103	22	34 242	42	62 325	62	79 239	82	91 381
3	47 712	23	36 173	43	63 347	63	79 934	83	91 908
4	60 206	24	38 021	44	64 345	64	80 618	84	92 428
5	69 897	25	39 794	45	65 321	65	81 291	85	92 942
6	77 815	26	41 497	46	66 276	66	81 954	86	93 450
7	84 510	27	43 136	47	67 210	67	82 607	87	93 952
8	90 309	28	44 716	48	68 124	68	83 251	88	94 448
9	95 424	29	46 240	49	69 020	69	83 885	89	94 939
10	00 000	30	47 712	50	69 897	70	84 510	90	95 424
11	04 139	31	49 136	51	70 757	71	85 126	91	95 904
12	07 918	32	50 515	52	71 600	72	85 733	92	96 379
13	11 394	33	51 851	53	72 428	73	86 332	93	96 848
14	14 613	34	53 148	54	73 239	74	86 923	94	97 313
15	17 609	35	54 407	55	74 036	75	87 506	95	97 772
16	20 412	36	55 630	56	74 819	76	88 081	96	98 227
17	23 045	37	56 820	57	75 587	77	88 649	97	98 677
18	25 527	38	57 978	58	76 343	78	89 209	98	99 123
19	27 875	39	59 106	59	77 085	79	89 763	99	99 564
20	30 103	40	60 206	60	77 815	80	90 309	100	00 000

COMMON LOGARITHMS.

N.	L. 0	1	2	3	4	5	6	7	8	9
100	00 000	043	087	130	173	217	260	303	346	389
101	432	475	518	561	604	647	689	732	775	817
102	860	903	945	988	*030	*072	*115	*157	*199	*242
103	01 284	326	368	410	452	494	536	578	620	662
104	703	745	787	828	870	912	953	995	*036	*078
105	02 119	160	202	243	284	325	366	407	449	490
106	531	572	612	653	694	735	776	816	857	898
107	938	979	*019	*060	*100	*141	*181	*222	*262	*302
108	03 342	383	423	463	503	543	583	623	663	703
109	743	782	822	862	902	941	981	*021	*060	*100
110	04 139	179	218	258	297	336	376	415	454	493
111	532	571	610	650	689	727	766	805	844	883
112	922	961	999	*038	*077	*115	*154	*192	*231	*269
113	05 308	346	385	423	461	500	538	576	614	652
114	690	729	767	805	843	881	918	956	994	*032
115	06 070	108	145	183	221	258	296	333	371	408
116	446	483	521	558	595	633	670	707	744	781
117	819	856	893	930	967	*004	*041	*078	*115	*151
118	07 188	225	262	298	335	372	408	445	482	518
119	555	591	628	664	700	737	773	809	846	882
120	918	954	990	*027	*063	*099	*135	*171	*207	*243
121	08 279	314	350	386	422	458	493	529	565	600
122	636	672	707	743	778	814	849	884	920	955
123	991	*026	*061	*096	*132	*167	*202	*237	*272	*307
124	09 342	377	412	447	482	517	552	587	621	656
125	691	726	760	795	830	864	899	934	968	*003
126	10 037	072	106	140	175	209	243	278	312	346
127	380	415	449	483	517	551	585	619	653	687
128	721	755	789	823	857	890	924	958	992	*025
129	11 059	093	126	160	193	227	261	294	327	361
130	394	428	461	494	528	561	594	628	661	694
131	727	760	793	826	860	893	926	959	992	*024
132	12 057	090	123	156	189	222	254	287	320	352
133	385	418	450	483	516	548	581	613	646	678
134	710	743	775	808	840	872	905	937	969	*001
135	13 033	066	098	130	162	194	226	258	290	322
136	354	386	418	450	481	513	545	577	609	640
137	672	704	735	767	799	830	862	893	925	956
138	988	*019	*051	*082	*114	*145	*176	*208	*239	*270
139	14 301	333	364	395	426	457	489	520	551	582
140	613	644	675	706	737	768	799	829	860	891
141	922	953	983	*014	*045	*076	*106	*137	*168	*198
142	15 229	259	290	320	351	381	412	442	473	503
143	534	564	594	625	655	685	715	746	776	806
144	836	866	897	927	957	987	*017	*047	*077	*107
145	16 137	167	197	227	256	286	316	346	376	406
146	435	465	495	524	554	584	613	643	673	702
147	732	761	791	820	850	879	909	938	967	997
148	17 026	056	085	114	143	173	202	231	260	289
149	319	348	377	406	435	464	493	522	551	580
150	609	638	667	696	725	754	782	811	840	869
N.	L. 0	1	2	3	4	5	6	7	8	9

P. P.

	44	43	42
1	4.4	4.3	4.2
2	8.8	8.6	8.4
3	13.2	12.9	12.6
4	17.6	17.2	16.8
5	22.0	21.5	21.0
6	26.4	25.8	25.2
7	30.8	30.1	29.4
8	35.2	34.4	33.6
9	39.6	38.7	37.8

	41	40	39
1	4.1	4.0	3.9
2	8.2	8.0	7.8
3	12.3	12.0	11.7
4	16.4	16.0	15.6
5	20.5	20.0	19.5
6	24.6	24.0	23.4
7	28.7	28.0	27.3
8	32.8	32.0	31.2
9	36.9	36.0	35.1

	38	37	36
1	3.8	3.7	3.6
2	7.6	7.4	7.2
3	11.4	11.1	10.8
4	15.2	14.8	14.4
5	19.0	18.5	18.0
6	22.8	22.2	21.6
7	26.6	25.9	25.2
8	30.4	29.6	28.8
9	34.2	33.3	32.4

	35	34	33
1	3.5	3.4	3.3
2	7.0	6.8	6.6
3	10.5	10.2	9.9
4	14.0	13.6	13.2
5	17.5	17.0	16.5
6	21.0	20.4	19.8
7	24.5	23.8	23.1
8	28.0	27.2	26.4
9	31.5	30.6	29.7

	32	31	30
1	3.2	3.1	3.0
2	6.4	6.2	6.0
3	9.6	9.3	9.0
4	12.8	12.4	12.0
5	16.0	15.5	15.0
6	19.2	18.6	18.0
7	22.4	21.7	21.0
8	25.6	24.8	24.0
9	28.8	27.9	27.0

P. P.

TABLE—(*Continued*).

N.	L. 0	1	2	3	4	5	6	7	8	9
150	17 609	638	667	696	725	754	782	811	840	869
151	898	926	955	984	*013	*041	*070	*099	*127	*156
152	18 184	213	241	270	298	327	355	384	412	441
153	469	498	526	554	583	611	639	667	696	724
154	752	780	808	837	865	893	921	949	977	*005
155	19 033	061	089	117	145	173	201	229	257	285
156	312	340	368	396	424	451	479	507	535	562
157	590	618	645	673	700	728	756	783	811	838
158	866	893	921	948	976	*003	*030	*058	*085	*112
159	20 140	167	194	222	249	276	303	330	358	385
160	412	439	466	493	520	548	575	602	629	656
161	683	710	737	763	790	817	844	871	898	925
162	952	978	*005	*032	*059	*085	*112	*139	*165	*192
163	21 219	245	272	299	325	352	378	405	431	458
164	484	511	537	564	590	617	643	669	696	722
165	748	775	801	827	854	880	906	932	958	985
166	22 011	037	063	089	115	141	167	194	220	246
167	272	298	324	350	376	401	427	453	479	505
168	531	557	583	608	634	660	686	712	737	763
169	789	814	840	866	891	917	943	968	994	*019
170	23 045	070	096	121	147	172	198	223	249	274
171	300	325	350	376	401	426	452	477	502	528
172	553	578	603	629	654	679	704	729	754	779
173	805	830	855	880	905	930	955	980	*005	*030
174	24 055	080	105	130	155	180	204	229	254	279
175	304	329	353	378	403	428	452	477	502	527
176	551	576	601	625	650	674	699	724	748	773
177	797	822	846	871	895	920	944	969	993	*018
178	25 042	066	091	115	139	164	188	212	237	261
179	285	310	334	358	382	406	431	455	479	503
180	527	551	575	600	624	648	672	696	720	744
181	768	792	816	840	864	888	912	935	959	983
182	26 007	031	055	079	102	126	150	174	198	221
183	245	269	293	316	340	364	387	411	435	458
184	482	505	529	553	576	600	623	647	670	694
185	717	741	764	788	811	834	858	881	905	928
186	951	975	998	*021	*045	*068	*091	*114	*138	*161
187	27 184	207	231	254	277	300	323	346	370	393
188	416	439	462	485	508	531	554	577	600	623
189	646	669	692	715	738	761	784	807	830	852
190	875	898	921	944	967	989	*012	*035	*058	*081
191	28 103	126	149	171	194	217	240	262	285	307
192	330	353	375	398	421	443	466	488	511	533
193	556	578	601	623	646	668	691	713	735	758
194	780	803	825	847	870	892	914	937	959	981
195	29 003	026	048	070	092	115	137	159	181	203
196	226	248	270	292	314	336	358	380	403	425
197	447	469	491	513	535	557	579	601	623	645
198	667	688	710	732	754	776	798	820	842	863
199	885	907	929	951	973	994	*016	*038	*060	*081
200	30 103	125	146	168	190	211	233	255	276	298
N.	L. 0	1	2	3	4	5	6	7	8	9

P. P.

	29	28
1	2.9	2.8
2	5.8	5.6
3	8.7	8.4
4	11.6	11.2
5	14.5	14.0
6	17.4	16.8
7	20.3	19.6
8	23.2	22.4
9	26.1	25.2

	27	26
1	2.7	2.6
2	5.4	5.2
3	8.1	7.8
4	10.8	10.4
5	13.5	13.0
6	16.2	15.6
7	18.9	18.2
8	21.6	20.8
9	24.3	23.4

	25
1	2.5
2	5.0
4	7.5
4	10.0
5	12.5
6	15.0
7	17.5
8	20.0
9	22.5

	24	23
1	2.4	2.3
2	4.8	4.6
3	7.2	6.9
4	9.6	9.2
5	12.0	11.5
6	14.4	13.8
7	16.8	16.1
8	19.2	18.4
9	21.6	20.7

	22	21
1	2.2	2.1
2	4.4	4.2
3	6.6	6.3
4	8.8	8.4
5	11.0	10.5
6	13.2	12.6
7	15.4	14.7
8	17.6	16.8
9	19.8	18.9

P. P.

TABLE—(*Continued*).

N.	L. 0	1	2	3	4	5	6	7	8	9
200	30 103	125	146	168	190	211	233	255	276	298
201	320	341	363	384	406	428	449	471	492	514
202	535	557	578	600	621	643	664	685	707	728
203	750	771	792	814	835	856	878	899	920	942
204	963	984	*006	*027	*048	*069	*091	*112	*133	*154
205	31 175	197	218	239	260	281	302	323	345	366
206	387	408	429	450	471	492	513	534	555	576
207	597	618	639	660	681	702	723	744	765	785
208	806	827	848	869	890	911	931	952	973	994
209	32 015	035	056	077	098	118	139	160	181	201
210	222	243	263	284	305	325	346	366	387	408
211	428	449	469	490	510	531	552	572	593	613
212	634	654	675	695	715	736	756	777	797	818
213	838	858	879	899	919	940	960	980	*001	*021
214	33 041	062	082	102	122	143	163	183	203	224
215	244	264	284	304	325	345	365	385	405	425
216	445	465	486	506	526	546	566	586	606	626
217	646	666	686	706	726	746	766	786	806	826
218	846	866	885	905	925	945	965	985	*005	*025
219	34 044	064	084	104	124	143	163	183	203	223
220	242	262	282	301	321	341	361	380	400	420
221	439	459	479	498	518	537	557	577	596	616
222	635	655	674	694	713	733	753	772	792	811
223	830	850	869	889	908	928	947	967	986	*005
224	35 025	044	064	083	102	122	141	160	180	199
225	218	238	257	276	295	315	334	353	372	392
226	411	430	449	468	488	507	526	545	564	583
227	603	622	641	660	679	698	717	736	755	774
228	793	813	832	851	870	889	908	927	946	965
229	984	*003	*021	*040	*059	*078	*097	*116	*135	*154
230	36 173	192	211	229	248	267	286	305	324	342
231	361	380	399	418	436	455	474	493	511	530
232	549	568	586	605	624	642	661	680	698	717
233	736	754	773	791	810	829	847	866	884	903
234	922	940	959	977	996	*014	*033	*051	*070	*088
235	37 107	125	144	162	181	199	218	236	254	273
236	291	310	328	346	365	383	401	420	438	457
237	475	493	511	530	548	566	585	603	621	639
238	658	676	694	712	731	749	767	785	803	822
239	840	858	876	894	912	931	949	967	985	*003
240	38 021	039	057	075	093	112	130	148	166	184
241	202	220	238	256	274	292	310	328	346	364
242	382	399	417	435	453	471	489	507	525	543
243	561	578	596	614	632	650	668	686	703	721
244	739	757	775	792	810	828	846	863	881	899
245	917	934	952	970	987	*005	*023	*041	*058	*076
246	39 094	111	129	146	164	182	199	217	235	252
247	270	287	305	322	340	358	375	393	410	428
248	445	463	480	498	515	533	550	568	585	602
249	620	637	655	672	690	707	724	742	759	777
250	794	811	829	846	863	881	898	915	933	950
N.	L. 0	1	2	3	4	5	6	7	8	9

P. P.

	22	21
1	2.2	2.1
2	4.4	4.2
3	6.6	6.3
4	8.8	8.4
5	11.0	10.5
6	13.2	12.6
7	15.4	14.7
8	17.6	16.8
9	19.8	18.9

	20
1	2.0
2	4.0
3	6.0
4	8.0
5	10.0
6	12.0
7	14.0
8	16.0
9	18.0

	19
1	1.9
2	3.8
3	5.7
4	7.6
5	9.5
6	11.4
7	13.3
8	15.2
9	17.1

	18
1	1.8
2	3.6
3	5.4
4	7.2
5	9.0
6	10.8
7	12.6
8	14.4
9	16.2

	17
1	1.7
2	3.4
3	5.1
4	6.8
5	8.5
6	10.2
7	11.9
8	13.6
9	15.3

P. P.

TABLE—(*Continued*).

N.	L. 0	1	2	3	4	5	6	7	8	9
250	39 794	811	829	846	863	881	898	915	933	950
251	967	985	*002	*019	*037	*054	*071	*088	*106	*123
252	40 140	157	175	192	209	226	243	261	278	295
253	312	329	346	364	381	398	415	432	449	466
254	483	500	518	535	552	569	586	603	620	637
255	654	671	688	705	722	739	756	773	790	807
256	824	841	858	875	892	909	926	943	960	976
257	993	*010	*027	*044	*061	*078	*095	*111	*128	*145
258	41 162	179	196	212	229	246	263	280	296	313
259	330	347	363	380	397	414	430	447	464	481
260	497	514	531	547	564	581	597	614	631	647
261	664	681	697	714	731	747	764	780	797	814
262	830	847	863	880	896	913	929	946	963	979
263	996	*012	*029	*045	*062	*078	*095	*111	*127	*144
264	42 160	177	193	210	226	243	259	275	292	308
265	325	341	357	374	390	406	423	439	455	472
266	488	504	521	537	553	570	586	602	619	635
267	651	667	684	700	716	732	749	765	781	797
268	813	830	846	862	878	894	911	927	943	959
269	975	991	*008	*024	*040	*056	*072	*088	*104	*120
270	43 136	152	169	185	201	217	233	249	265	281
271	297	313	329	345	361	377	393	409	425	441
272	457	473	489	505	521	537	553	569	584	600
273	616	632	648	664	680	696	712	727	743	759
274	775	791	807	823	838	854	870	886	902	917
275	933	949	965	981	996	*012	*028	*044	*059	*075
276	44 091	107	122	138	154	170	185	201	217	232
277	248	264	279	295	311	326	342	358	373	389
278	404	420	436	451	467	483	498	514	529	545
279	560	576	592	607	623	638	654	669	685	700
280	716	731	747	762	778	793	809	824	840	855
281	871	886	902	917	932	948	963	979	994	*010
282	45 025	040	056	071	086	102	117	133	148	163
283	179	194	209	225	240	255	271	286	301	317
284	332	347	362	378	393	408	423	439	454	469
285	484	500	515	530	545	561	576	591	606	621
286	637	652	667	682	697	712	728	743	758	773
287	788	803	818	834	849	864	879	894	909	924
288	939	954	969	984	*000	*015	*030	*045	*060	*075
289	46 090	105	120	135	150	165	180	195	210	225
290	240	255	270	285	300	315	330	345	359	374
291	389	404	419	434	449	464	479	494	509	523
292	538	553	568	583	598	613	627	642	657	672
293	687	702	716	731	746	761	776	790	805	820
294	835	850	864	879	894	909	923	938	953	967
295	982	997	*012	*026	*041	*056	*070	*085	*100	*114
296	47 129	144	159	173	188	202	217	232	246	261
297	276	290	305	319	334	349	363	378	392	407
298	422	436	451	465	480	494	509	524	538	553
299	567	582	596	611	625	640	654	669	683	698
300	712	727	741	756	770	784	799	813	828	842
N.	L. 0	1	2	3	4	5	6	7	8	9

P. P.

	18	17	16	15	14
1	1.8	1.7	1.6	1.5	1.4
2	3.6	3.4	3.2	3.0	2.8
3	5.4	5.1	4.8	4.5	4.2
4	7.2	6.8	6.4	6.0	5.6
5	9.0	8.5	8.0	7.5	7.0
6	10.8	10.2	9.6	9.0	8.4
7	12.6	11.9	11.2	10.5	9.8
8	14.4	13.6	12.8	12.0	11.2
9	16.2	15.3	14.4	13.5	12.6

P. P.

TABLE—(*Continued*).

N.	L. 0	1	2	3	4	5	6	7	8	9
300	47 712	727	741	756	770	784	799	813	828	842
301	857	871	885	900	914	929	943	958	972	986
302	48 001	015	029	044	058	073	087	101	116	130
303	144	159	173	187	202	216	230	244	259	273
304	287	302	316	330	344	359	373	387	401	416
305	430	444	458	473	487	501	515	530	544	558
306	572	586	601	615	629	643	657	671	686	700
307	714	728	742	756	770	785	799	813	827	841
308	855	869	883	897	911	926	940	954	968	982
309	996	*010	*024	*038	*052	*066	*080	*094	*108	*122
310	49 136	150	164	178	192	206	220	234	248	262
311	276	290	304	318	332	346	360	374	388	402
312	415	429	443	457	471	485	499	513	527	541
313	554	568	582	596	610	624	638	651	665	679
314	693	707	721	734	748	762	776	790	803	817
315	831	845	859	872	886	900	914	927	941	955
316	969	982	996	*010	*024	*037	*051	*065	*079	*092
317	50 106	120	133	147	161	174	188	202	215	229
318	243	256	270	284	297	311	325	338	352	365
319	379	393	406	420	433	447	461	474	488	501
320	515	529	542	556	569	583	596	610	623	637
321	651	664	678	691	705	718	732	745	759	772
322	786	799	813	826	840	853	866	880	893	907
323	920	934	947	961	974	987	*001	*014	*028	*041
324	51 055	068	081	095	108	121	135	148	162	175
325	188	202	215	228	242	255	268	282	295	308
326	322	335	348	362	375	388	402	415	428	441
327	455	468	481	495	508	521	534	548	561	574
328	587	601	614	627	640	654	667	680	693	706
329	720	733	746	759	772	786	799	812	825	838
330	851	865	878	891	904	917	930	943	957	970
331	983	996	*009	*022	*035	*048	*061	*075	*088	*101
332	52 114	127	140	153	166	179	192	205	218	231
333	244	257	270	284	297	310	323	336	349	362
334	375	388	401	414	427	440	453	466	479	492
335	504	517	530	543	556	569	582	595	608	621
336	634	647	660	673	686	699	711	724	737	750
337	763	776	789	802	815	827	840	853	866	879
338	892	905	917	930	943	956	969	982	994	*007
339	53 020	033	046	058	071	084	097	110	122	135
340	148	161	173	186	199	212	224	237	250	263
341	275	288	301	314	326	339	352	364	377	390
342	403	415	428	441	453	466	479	491	504	517
343	529	542	555	567	580	593	605	618	631	643
344	656	668	681	694	706	719	732	744	757	769
345	782	794	807	820	832	845	857	870	882	895
346	908	920	933	945	958	970	983	995	*008	*020
347	54 033	045	058	070	083	095	108	120	133	145
348	158	170	183	195	208	220	233	245	258	270
349	283	295	307	320	332	345	357	370	382	394
350	407	419	432	444	456	469	481	494	506	518
N.	L. 0	1	2	3	4	5	6	7	8	9

P. P.

	15
1	1.5
2	3.0
3	4.5
4	6.0
5	7.5
6	9.0
7	10.5
8	12.0
9	13.5

	14
1	1.4
2	2.8
3	4.2
4	5.6
5	7.0
6	8.4
7	9.8
8	11.2
9	12.6

	13
1	1.3
2	2.6
3	3.9
4	5.2
5	6.5
6	7.8
7	9.1
8	10.4
9	11.7

	12
1	1.2
2	2.4
3	3.6
4	4.8
5	6.0
6	7.2
7	8.4
8	9.6
9	10.8

P. P.

TABLE—(*Continued*).

N.	L. 0	1	2	3	4	5	6	7	8	9
350	54 407	419	432	444	456	469	481	494	506	518
351	531	543	555	568	580	593	605	617	630	642
352	654	667	679	691	704	716	728	741	753	765
353	777	790	802	814	827	839	851	864	876	888
354	900	913	925	937	949	962	974	986	998	*011
355	55 023	035	047	060	072	084	096	108	121	133
356	145	157	169	182	194	206	218	230	242	255
357	267	279	291	303	315	328	340	352	364	376
358	388	400	413	425	437	449	461	473	485	497
359	509	522	534	546	558	570	582	594	606	618
360	630	642	654	666	678	691	703	715	727	739
361	751	763	775	787	799	811	823	835	847	859
362	871	883	895	907	919	931	943	955	967	979
363	991	*003	*015	*027	*038	*050	*062	*074	*086	*098
364	56 110	122	134	146	158	170	182	194	205	217
365	229	241	253	265	277	289	301	312	324	336
366	348	360	372	384	396	407	419	431	443	455
367	467	478	490	502	514	526	538	549	561	573
368	585	597	608	620	632	644	656	667	679	691
369	703	714	726	738	750	761	773	785	797	808
370	820	832	844	855	867	879	891	902	914	926
371	937	949	961	972	984	996	*008	*019	*031	*043
372	57 054	066	078	089	101	113	124	136	148	159
373	171	183	194	206	217	229	241	252	264	276
374	287	299	310	322	334	345	357	368	380	392
375	403	415	426	438	449	461	473	484	496	507
376	519	530	542	553	565	576	588	600	611	623
377	634	646	657	669	680	692	703	715	726	738
378	749	761	772	784	795	807	818	830	841	852
379	864	875	887	898	910	921	933	944	955	967
380	978	990	*001	*013	*024	*035	*047	*058	*070	*081
381	58 092	104	115	127	138	149	161	172	184	195
382	206	218	229	240	252	263	274	286	297	309
383	320	331	343	354	365	377	388	399	410	422
384	433	444	456	467	478	490	501	512	524	535
385	546	557	569	580	591	602	614	625	636	647
386	659	670	681	692	704	715	726	737	749	760
387	771	782	794	805	816	827	838	850	861	872
388	883	894	906	917	928	939	950	961	973	984
389	995	*006	*017	*028	*040	*051	*062	*073	*084	*095
390	59 106	118	129	140	151	162	173	184	195	207
391	218	229	240	251	262	273	284	295	306	318
392	329	340	351	362	373	384	395	406	417	428
393	439	450	461	472	483	494	506	517	528	539
394	550	561	572	583	594	605	616	627	638	649
395	660	671	682	693	704	715	726	737	748	759
396	770	780	791	802	813	824	835	846	857	868
397	879	890	901	912	923	934	945	956	966	977
398	988	999	*010	*021	*032	*043	*054	*065	*076	*086
399	60 097	108	119	130	141	152	163	173	184	195
400	206	217	228	239	249	260	271	282	293	304
N.	L. 0	1	2	3	4	5	6	7	8	9

P. P.

	13
1	1.3
2	2.6
3	3.9
4	5.2
5	6.5
6	7.8
7	9.1
8	10.4
9	11.7

	12
1	1.2
2	2.4
3	3.6
4	4.8
5	6.0
6	7.2
7	8.4
8	9.6
9	10.8

	11
1	1.1
2	2.2
3	3.3
4	4.4
5	5.5
6	6.6
7	7.7
8	8.8
9	9.9

	10
1	1.0
2	2.0
3	3.0
4	4.0
5	5.0
6	6.0
7	7.0
8	8.0
9	9.0

P. P.

TABLE—(*Continued*).

N.	L. 0	1	2	3	4	5	6	7	8	9
400	60 206	217	228	239	249	260	271	282	293	304
401	314	325	336	347	358	369	379	390	401	412
402	423	433	444	455	466	477	487	498	509	520
403	531	541	552	563	574	584	595	606	617	627
404	638	649	660	670	681	692	703	713	724	735
405	746	756	767	778	788	799	810	821	831	842
406	853	863	874	885	895	906	917	927	938	949
407	959	970	981	991	*002	*013	*023	*034	*045	*055
408	61 066	077	087	098	109	119	130	140	151	162
409	172	183	194	204	215	225	236	247	257	268
410	278	289	300	310	321	331	342	352	363	374
411	384	395	405	416	426	437	448	458	469	479
412	490	500	511	521	532	542	553	563	574	584
413	595	606	616	627	637	648	658	669	679	690
414	700	711	721	731	742	752	763	773	784	794
415	805	815	826	836	847	857	868	878	888	899
416	909	920	930	941	951	962	972	982	993	*003
417	62 014	024	034	045	055	066	076	086	097	107
418	118	128	138	149	159	170	180	190	201	211
419	221	232	242	252	263	273	284	294	304	315
420	325	335	346	356	366	377	387	397	408	418
421	428	439	449	459	469	480	490	500	511	521
422	531	542	552	562	572	583	593	603	613	624
423	634	644	655	665	675	685	696	706	716	726
424	737	747	757	767	778	788	798	808	818	829
425	839	849	859	870	880	890	900	910	921	931
426	941	951	961	972	982	992	*002	*012	*022	*033
427	63 043	053	063	073	083	094	104	114	124	134
428	144	155	165	175	185	195	205	215	225	236
429	246	256	266	276	286	296	306	317	327	337
430	347	357	367	377	387	397	407	417	428	438
431	448	458	468	478	488	498	508	518	528	538
432	548	558	568	579	589	599	609	619	629	639
433	649	659	669	679	689	699	709	719	729	739
434	749	759	769	779	789	799	809	819	829	839
435	849	859	869	879	889	899	909	919	929	939
436	949	959	969	979	988	998	*008	*018	*028	*038
437	64 048	058	068	078	088	098	108	118	128	137
438	147	157	167	177	187	197	207	217	227	237
439	246	256	266	276	286	296	306	316	326	335
440	345	355	365	375	385	395	404	414	424	434
441	444	454	464	473	483	493	503	513	523	532
442	542	552	562	572	582	591	601	611	621	631
443	640	650	660	670	680	689	699	709	719	729
444	738	748	758	768	777	787	797	807	816	826
445	836	846	856	865	875	885	895	904	914	924
446	933	943	953	963	972	982	992	*002	*011	*021
447	65 031	040	050	060	070	079	089	099	108	118
448	128	137	147	157	167	176	186	196	205	215
449	225	234	244	254	263	273	283	292	302	312
450	321	331	341	350	360	369	379	389	398	408
N.	L. 0	1	2	3	4	5	6	7	8	9

P. P.

	11
1	1.1
2	2.2
3	3.3
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6	6.6
7	7.7
8	8.8
9	9.9

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7	7.0
8	8.0
9	9.0

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2	1.8
3	2.7
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6	5.4
7	6.3
8	7.2
9	8.1

P. P.

TABLE—(*Continued*).

N.	L. 0	1	2	3	4	5	6	7	8	9
450	65 321	331	341	350	360	369	379	389	398	408
451	418	427	437	447	456	466	475	485	495	504
452	514	523	533	543	552	562	571	581	591	600
453	610	619	629	639	648	658	667	677	686	696
454	706	715	725	734	744	753	763	772	782	792
455	801	811	820	830	839	849	858	868	877	887
456	896	906	916	925	935	944	954	963	973	982
457	992	*001	*011	*020	*030	*039	*049	*058	*068	*077
458	66 087	096	106	115	124	134	143	153	162	172
459	181	191	200	210	219	229	238	247	257	266
460	276	285	295	304	314	323	332	342	351	361
461	370	380	389	398	408	417	427	436	445	455
462	464	474	483	492	502	511	521	530	539	549
463	558	567	577	586	596	605	614	624	633	642
464	652	661	671	680	689	699	708	717	727	736
465	745	755	764	773	783	792	801	811	820	829
466	839	848	857	867	876	885	894	904	913	922
467	932	941	950	960	969	978	987	997	*006	*015
468	67 025	034	043	052	062	071	080	089	099	108
469	117	127	136	145	154	164	173	182	191	201
470	210	219	228	237	247	256	265	274	284	293
471	302	311	321	330	339	348	357	367	376	385
472	394	403	413	422	431	440	449	459	468	477
473	486	495	504	514	523	532	541	550	560	569
474	578	587	596	605	614	624	633	642	651	660
475	669	679	688	697	706	715	724	733	742	752
476	761	770	779	788	797	806	815	825	834	843
477	852	861	870	879	888	897	906	916	925	934
478	943	952	961	970	979	988	997	*006	*015	*024
479	68 034	043	052	061	070	079	088	097	106	115
480	124	133	142	151	160	169	178	187	196	205
481	215	224	233	242	251	260	269	278	287	296
482	305	314	323	332	341	350	359	368	377	386
483	395	404	413	422	431	440	449	458	467	476
484	485	494	502	511	520	529	538	547	556	565
485	574	583	592	601	610	619	628	637	646	655
486	664	673	681	690	699	708	717	726	735	744
487	753	762	771	780	789	797	806	815	824	833
488	842	851	860	869	878	886	895	904	913	922
489	931	940	949	958	966	975	984	993	*002	*011
490	69 020	028	037	046	055	064	073	082	090	099
491	108	117	126	135	144	152	161	170	179	188
492	197	205	214	223	232	241	249	258	267	276
493	285	294	302	311	320	329	338	346	355	364
494	373	381	390	399	408	417	425	434	443	452
495	461	469	478	487	496	504	513	522	531	539
496	548	557	566	574	583	592	601	609	618	627
497	636	644	653	662	671	679	688	697	705	714
498	723	732	740	749	758	767	775	784	793	801
499	810	819	827	836	845	854	862	871	880	888
500	897	906	914	923	932	940	949	958	966	975
N.	L. 0	1	2	3	4	5	6	7	8	9

P. P.

	10
1	1.0
2	2.0
3	3.0
4	4.0
5	5.0
6	6.0
7	7.0
8	8.0
9	9.0

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1	0.9
2	1.8
3	2.7
4	3.6
5	4.5
6	5.4
7	6.3
8	7.2
9	8.1

	8
1	0.8
2	1.6
3	2.4
4	3.2
5	4.0
6	4.8
7	5.6
8	6.4
9	7.2

P. P.

TABLE—(*Continued*).

N.	L. 0	1	2	3	4	5	6	7	8	9
500	69 897	906	914	923	932	940	949	958	966	975
501	984	992	*001	*010	*018	*027	*036	*044	*053	*062
502	70 070	079	088	096	105	114	122	131	140	148
503	157	165	174	183	191	200	209	217	226	234
504	243	252	260	269	278	286	295	303	312	321
505	329	338	346	355	364	372	381	389	398	406
506	415	424	432	441	449	458	467	475	484	492
507	501	509	518	526	535	544	552	561	569	578
508	586	595	603	612	621	629	638	646	655	663
509	672	680	689	697	706	714	723	731	740	749
510	757	766	774	783	791	800	808	817	825	834
511	842	851	859	868	876	885	893	902	910	919
512	927	935	944	952	961	969	978	986	995	*003
513	71 012	020	029	037	046	054	063	071	079	088
514	096	105	113	122	130	139	147	155	164	172
515	181	189	198	206	214	223	231	240	248	257
516	265	273	282	290	299	307	315	324	332	341
517	349	357	366	374	383	391	399	408	416	425
518	433	441	450	458	466	475	483	492	500	508
519	517	525	533	542	550	559	567	575	584	592
520	600	609	617	625	634	642	650	659	667	675
521	684	692	700	709	717	725	734	742	750	759
522	767	775	784	792	800	809	817	825	834	842
523	850	858	867	875	883	892	900	908	917	925
524	933	941	950	958	966	975	983	991	999	*008
525	72 016	024	032	041	049	057	066	074	082	090
526	099	107	115	123	132	140	148	156	165	173
527	181	189	198	206	214	222	230	239	247	255
528	263	272	280	288	296	304	313	321	329	337
529	346	354	362	370	378	387	395	403	411	419
530	428	436	444	452	460	469	477	485	493	501
531	509	518	526	534	542	550	558	567	575	583
532	591	599	607	616	624	632	640	648	656	665
533	673	681	689	697	705	713	722	730	738	746
534	754	762	770	779	787	795	803	811	819	827
535	835	843	852	860	868	876	884	892	900	908
536	916	925	933	941	949	957	965	973	981	989
537	997	*006	*014	*022	*030	*038	*046	*054	*062	*070
538	73 078	086	094	102	111	119	127	135	143	151
539	159	167	175	183	191	199	207	215	223	231
540	239	247	255	263	272	280	288	296	304	312
541	320	328	336	344	352	360	368	376	384	392
542	400	408	416	424	432	440	448	456	464	472
543	480	488	496	504	512	520	528	536	544	552
544	560	568	576	584	592	600	608	616	624	632
545	640	648	656	664	672	679	687	695	703	711
546	719	727	735	743	751	759	767	775	783	791
547	799	807	815	823	830	838	846	854	862	870
548	878	886	894	902	910	918	926	933	941	949
549	957	965	973	981	989	997	*005	*013	*020	*028
550	74 036	044	052	060	068	076	084	092	099	107
N.	L. 0	1	2	3	4	5	6	7	8	9

P. P.

	9
1	0.9
2	1.8
3	2.7
4	3.6
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6	5.4
7	6.3
8	7.2
9	8.1

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2	1.6
3	2.4
4	3.2
5	4.0
6	4.8
7	5.6
8	6.4
9	7.2

	7
1	0.7
2	1.4
3	2.1
4	2.8
5	3.5
6	4.2
7	4.9
8	5.6
9	6.3

P. P.

TABLE—(*Continued*).

N.	L. 0	1	2	3	4	5	6	7	8	9	P. P.
550	74 036	044	052	060	068	076	084	092	099	107	
551	115	123	131	139	147	155	162	170	178	186	
552	194	202	210	218	225	233	241	249	257	265	
553	273	280	288	296	304	312	320	327	335	343	
554	351	359	367	374	382	390	398	406	414	421	
555	429	437	445	453	461	468	476	484	492	500	
556	507	515	523	531	539	547	554	562	570	578	
557	586	593	601	609	617	624	632	640	648	656	
558	663	671	679	687	695	702	710	718	726	733	
559	741	749	757	764	772	780	788	796	803	811	
560	819	827	834	842	850	858	865	873	881	889	
561	896	904	912	920	927	935	943	950	958	966	**8**
562	974	981	989	997	*005	*012	*020	*028	*035	*043	1 | 0.8
563	75 051	059	066	074	082	089	097	105	113	120	2 | 1.6
564	128	136	143	151	159	166	174	182	189	197	3 | 2.4
565	205	213	220	228	236	243	251	259	266	274	4 | 3.2
566	282	289	297	305	312	320	328	335	343	351	5 | 4.0
567	358	366	374	381	289	397	404	412	420	427	6 | 4.8
568	435	442	450	458	465	473	481	488	496	504	7 | 5.6
569	511	519	526	534	542	549	557	565	572	580	8 | 6.4
570	587	595	603	610	618	626	633	641	648	656	9 | 7.2
571	664	671	679	686	694	702	709	717	724	732	
572	740	747	755	762	770	778	785	793	800	808	
573	815	823	831	838	846	853	861	868	876	884	
574	891	899	906	914	921	929	937	944	952	959	
575	967	974	982	989	997	*005	*012	*020	*027	*035	
576	76 042	050	057	065	072	080	087	095	103	110	
577	118	125	133	140	148	155	163	170	178	185	
578	193	200	208	215	223	230	238	245	253	260	
579	268	275	283	290	298	305	313	320	328	335	
580	343	350	358	365	373	380	388	395	403	410	
581	418	425	433	440	448	455	462	470	477	485	**7**
582	492	500	507	515	522	530	537	545	552	559	1 | 0.7
583	567	574	582	589	597	604	612	619	626	634	2 | 1.4
584	641	649	656	664	671	678	686	693	701	708	3 | 2.1
585	716	723	730	738	745	753	760	768	775	782	4 | 2.8
586	790	797	805	812	819	827	834	842	849	856	5 | 3.5
587	864	871	879	886	893	901	908	916	923	930	6 | 4.2
588	938	945	953	960	967	975	982	989	997	*004	7 | 4.9
589	77 012	019	026	034	041	048	056	063	070	078	8 | 5.6
590	085	093	100	107	115	122	129	137	144	151	9 | 6.3
591	159	166	173	181	188	195	203	210	217	225	
592	232	240	247	254	262	269	276	283	291	298	
593	305	313	320	327	335	342	349	357	364	371	
594	379	386	393	401	408	415	422	430	437	444	
595	452	459	466	474	481	488	495	503	510	517	
596	525	532	539	546	554	561	568	576	583	590	
597	597	605	612	619	627	634	641	648	656	663	
598	670	677	685	692	699	706	714	721	728	735	
599	743	750	757	764	772	779	786	793	801	808	
600	815	822	830	837	844	851	859	866	873	880	
N.	L. 0	1	2	3	4	5	6	7	8	9	P. P.

TABLE—(*Continued*).

N.	L. 0	1	2	3	4	5	6	7	8	9
600	77 815	822	830	837	844	851	859	866	873	880
601	887	895	902	909	916	924	931	938	945	952
602	960	967	974	981	988	996	*003	*010	*017	*025
603	78 032	039	046	053	061	068	075	082	089	097
604	104	111	118	125	132	140	147	154	161	168
605	176	183	190	197	204	211	219	226	233	240
606	247	254	262	269	276	283	290	297	305	312
607	319	326	333	340	347	355	362	369	376	383
608	390	398	405	412	419	426	433	440	447	455
609	462	469	476	483	490	497	504	512	519	526
610	533	540	547	554	561	569	576	583	590	597
611	604	611	618	625	633	640	647	654	661	668
612	675	682	689	696	704	711	718	725	732	739
613	746	753	760	767	774	781	789	796	803	810
614	817	824	831	838	845	852	859	866	873	880
615	888	895	902	909	916	923	930	937	944	951
616	958	965	972	979	986	993	*000	*007	*014	*021
617	79 029	036	043	050	057	064	071	078	085	092
618	099	106	113	120	127	134	141	148	155	162
619	169	176	183	190	197	204	211	218	225	232
620	239	246	253	260	267	274	281	288	295	302
621	309	316	323	330	337	344	351	358	365	372
622	379	386	393	400	407	414	421	428	435	442
623	449	456	463	470	477	484	491	498	505	511
624	518	525	532	539	546	553	560	567	574	581
625	588	595	602	609	616	623	630	637	644	650
626	657	664	671	678	685	692	699	706	713	720
627	727	734	741	748	754	761	768	775	782	789
628	796	803	810	817	824	831	837	844	851	858
629	865	872	879	886	893	900	906	913	920	927
630	934	941	948	955	962	969	975	982	989	996
631	80 003	010	017	024	030	037	044	051	058	065
632	072	079	085	092	099	106	113	120	127	134
633	140	147	154	161	168	175	182	188	195	202
634	209	216	223	229	236	243	250	257	264	271
635	277	284	291	298	305	312	318	325	332	339
636	346	353	359	366	373	380	387	393	400	407
637	414	421	428	434	441	448	455	462	468	475
638	482	489	496	502	509	516	523	530	536	543
639	550	557	564	570	577	584	591	598	604	611
640	618	625	632	638	645	652	659	665	672	679
641	686	693	699	706	713	720	726	733	740	747
642	754	760	767	774	781	787	794	801	808	814
643	821	828	835	841	848	855	862	868	875	882
644	889	895	902	909	916	922	929	936	943	949
645	956	963	969	976	983	990	996	*003	*010	*017
646	81 023	030	037	043	050	057	064	070	077	084
647	090	097	104	111	117	124	131	137	144	151
648	158	164	171	178	184	191	198	204	211	218
649	224	231	238	245	251	258	265	271	278	285
650	291	298	305	311	318	325	331	338	345	351
N.	L. 0	1	2	3	4	5	6	7	8	9

P. P.

	8
1	0.8
2	1.6
3	2.4
4	3.2
5	4.0
6	4.8
7	5.6
8	6.4
9	7.2

	7
1	0.7
2	1.4
3	2.1
4	2.8
5	3.5
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7	4.9
8	5.6
9	6.3

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2	1.2
3	1.8
4	2.4
5	3.0
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7	4.2
8	4.8
9	5.4

P. P.

TABLE—(*Continued*).

N.	L. 0	1	2	3	4	5	6	7	8	9
650	81 291	298	305	311	318	325	331	338	345	351
651	358	365	371	378	385	391	398	405	411	418
652	425	431	438	445	451	458	465	471	478	485
653	491	498	505	511	518	525	531	538	544	551
654	558	564	571	578	584	591	598	604	611	617
655	624	631	637	644	651	657	664	671	677	684
656	690	697	704	710	717	723	730	737	743	750
657	757	763	770	776	783	790	796	803	809	816
658	823	829	836	842	849	856	862	869	875	882
659	889	895	902	908	915	921	928	935	941	948
660	954	961	968	974	981	987	994	*000	*007	*014
661	82 020	027	033	040	046	053	060	066	073	079
662	086	092	099	105	112	119	125	132	138	145
663	151	158	164	171	178	184	191	197	204	210
664	217	223	230	236	243	249	256	263	269	276
665	282	289	295	302	308	315	321	328	334	341
666	347	354	360	367	373	380	387	393	400	406
667	413	419	426	432	439	445	452	458	465	471
668	478	484	491	497	504	510	517	523	530	536
669	543	549	556	562	569	575	582	588	595	601
670	607	614	620	627	633	640	646	653	659	666
671	672	679	685	692	698	705	711	718	724	730
672	737	743	750	756	763	769	776	782	789	795
673	802	808	814	821	827	834	840	847	853	860
674	866	872	879	885	892	898	905	911	918	924
675	930	937	943	950	956	963	969	975	982	988
676	995	*001	*008	*014	*020	*027	*033	*040	*046	*052
677	83 059	065	072	078	085	091	097	104	110	117
678	123	129	136	142	149	155	161	168	174	181
679	187	193	200	206	213	219	225	232	238	245
680	251	257	264	270	276	283	289	296	302	308
681	315	321	327	334	340	347	353	359	366	372
682	378	385	391	398	404	410	417	423	429	436
683	442	448	455	461	467	474	480	487	493	499
684	506	512	518	525	531	537	544	550	556	563
685	569	575	582	588	594	601	607	613	620	626
686	632	639	645	651	658	664	670	677	683	689
687	696	702	708	715	721	727	734	740	746	753
688	759	765	771	778	784	790	797	803	809	816
689	822	828	835	841	847	853	860	866	872	879
690	885	891	897	904	910	916	923	929	935	942
691	948	954	960	967	973	979	985	992	998	*004
692	84 011	017	023	029	036	042	048	055	061	067
693	073	080	086	092	098	105	111	117	123	130
694	136	142	148	155	161	167	173	180	186	192
695	198	205	211	217	223	230	236	242	248	255
696	261	267	273	280	286	292	298	305	311	317
697	323	330	336	342	348	354	361	367	373	379
698	386	392	398	404	410	417	423	429	435	442
699	448	454	460	466	473	479	485	491	497	504
700	510	516	522	528	535	541	547	553	559	566
N.	L. 0	1	2	3	4	5	6	7	8	9

P. P.

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7	4.9
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9	6.3

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7	4.2
8	4.8
9	5.4

P. P.

TABLE—(*Continued*).

N.	L. 0	1	2	3	4	5	6	7	8	9
700	84 510	516	522	528	535	541	547	553	559	566
701	572	578	584	590	597	603	609	615	621	628
702	634	640	646	652	658	665	671	677	683	689
703	696	702	708	714	720	726	733	739	745	751
704	757	763	770	776	782	788	794	800	807	813
705	819	825	831	837	844	850	856	862	868	874
706	880	887	893	899	905	911	917	924	930	936
707	942	948	954	960	967	973	979	985	991	997
708	85 003	009	016	022	028	034	040	046	052	058
709	065	071	077	083	089	095	101	107	114	120
710	126	132	138	144	150	156	163	169	175	181
711	187	193	199	205	211	217	224	230	236	242
712	248	254	260	266	272	278	285	291	297	303
713	309	315	321	327	333	339	345	352	358	364
714	370	376	382	388	394	400	406	412	418	425
715	431	437	443	449	455	461	467	473	479	485
716	491	497	503	509	516	522	528	534	540	546
717	552	558	564	570	576	582	588	594	600	606
718	612	618	625	631	637	643	649	655	661	667
719	673	679	685	691	697	703	709	715	721	727
720	733	739	745	751	757	763	769	775	781	788
721	794	800	806	812	818	824	830	836	842	848
722	854	860	866	872	878	884	890	896	902	908
723	914	920	926	932	938	944	950	956	962	968
724	974	980	986	992	998	*004	*010	*016	*022	*028
725	86 034	040	046	052	058	064	070	076	082	088
726	094	100	106	112	118	124	130	136	141	147
727	153	159	165	171	177	183	189	195	201	207
728	213	219	225	231	237	243	249	255	261	267
729	273	279	285	291	297	303	308	314	320	326
730	332	338	344	350	356	362	368	374	380	386
731	392	398	404	410	415	421	427	433	439	445
732	451	457	463	469	475	481	487	493	499	504
733	510	516	522	528	534	540	546	552	558	564
734	570	576	581	587	593	599	605	611	617	623
735	629	635	641	646	652	658	664	670	676	682
736	688	694	700	705	711	717	723	729	735	741
737	747	753	759	764	770	776	782	788	794	800
738	806	812	817	823	829	835	841	847	853	859
739	864	870	876	882	888	894	900	906	911	917
740	923	929	935	941	947	953	958	964	970	976
741	982	988	994	999	*005	*011	*017	*023	*029	*035
742	87 040	046	052	058	064	070	075	081	087	093
743	099	105	111	116	122	128	134	140	146	151
744	157	163	169	175	181	186	192	198	204	210
745	216	221	227	233	239	245	251	256	262	268
746	274	280	286	291	297	303	309	315	320	326
747	332	338	344	349	355	361	367	373	379	384
748	390	396	402	408	413	419	425	431	437	442
749	448	454	460	466	471	477	483	489	495	500
750	506	512	518	523	529	535	541	547	552	558
N.	L. 0	1	2	3	4	5	6	7	8	9

P. P.

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9	6.3

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6	3.6
7	4.2
8	4.8
9	5.4

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8	4.0
9	4.5

P. P.

TABLE—(*Continued*).

N.	L. 0	1	2	3	4	5	6	7	8	9
750	87 506	512	518	523	529	535	541	547	552	558
751	564	570	576	581	587	593	599	604	610	616
752	622	628	633	639	645	651	656	662	668	674
753	679	685	691	697	703	708	714	720	726	731
754	737	743	749	754	760	766	772	777	783	789
755	795	800	806	812	818	823	829	835	841	846
756	852	858	864	869	875	881	887	892	898	904
757	910	915	921	927	933	938	944	950	955	961
758	967	973	978	984	990	996	*001	*007	*013	*018
759	88 024	030	036	041	047	053	058	064	070	076
760	081	087	093	098	104	110	116	121	127	133
761	138	144	150	156	161	167	173	178	184	190
762	195	201	207	213	218	224	230	235	241	247
763	252	258	264	270	275	281	287	292	298	304
764	309	315	321	326	332	338	343	349	355	360
765	366	372	377	383	389	395	400	406	412	417
766	423	429	434	440	446	451	457	463	468	474
767	480	485	491	497	502	508	513	519	525	530
768	536	542	547	553	559	564	570	576	581	587
769	593	598	604	610	615	621	627	632	638	643
770	649	655	660	666	672	677	683	689	694	700
771	705	711	717	722	728	734	739	745	750	756
772	762	767	773	779	784	790	795	801	807	812
773	818	824	829	835	840	846	852	857	863	868
774	874	880	885	891	897	902	908	913	919	925
775	930	936	941	947	953	958	964	969	975	981
776	986	992	997	*003	*009	*014	*020	*025	*031	*037
777	89 042	048	053	059	064	070	076	081	087	092
778	098	104	109	115	120	126	131	137	143	148
779	154	159	165	170	176	182	187	193	198	204
780	209	215	221	226	232	237	243	248	254	260
781	265	271	276	282	287	293	298	304	310	315
782	321	326	332	337	343	348	354	360	365	371
783	376	382	387	393	398	404	409	415	421	426
784	432	437	443	448	454	459	465	470	476	481
785	487	492	498	504	509	515	520	526	531	537
786	542	548	553	559	564	570	575	581	586	592
787	597	603	609	614	620	625	631	636	642	647
788	653	658	664	669	675	680	686	691	697	702
789	708	713	719	724	730	735	741	746	752	757
790	763	768	774	779	785	790	796	801	807	812
791	818	823	829	834	840	845	851	856	862	867
792	873	878	883	889	894	900	905	911	916	922
793	927	933	938	944	949	955	960	966	971	977
794	982	988	993	998	*004	*009	*015	*020	*026	*031
795	90 037	042	048	053	059	064	069	075	080	086
796	091	097	102	108	113	119	124	129	135	140
797	146	151	157	162	168	173	179	184	189	195
798	200	206	211	217	222	227	233	238	244	249
799	255	260	266	271	276	282	287	293	298	304
800	309	314	320	325	331	336	342	347	352	358
N.	L. 0	1	2	3	4	5	6	7	8	9

P. P.

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1	0.6
2	1.2
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7	4.2
8	4.8
9	5.4

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3	1.5
4	2.0
5	2.5
6	3.0
7	3.5
8	4.0
9	4.5

P. P.

TABLE—(*Continued*).

N.	L. 0	1	2	3	4	5	6	7	8	9	P. P.
800	90 309	314	320	325	331	336	342	347	352	358	
801	363	369	374	380	385	390	396	401	407	412	
802	417	423	428	434	439	445	450	455	461	466	
803	472	477	482	488	493	499	504	509	515	520	
804	526	531	536	542	547	553	558	563	569	574	
805	580	585	590	596	601	607	612	617	623	628	
806	634	639	644	650	655	660	666	671	677	682	
807	687	693	698	703	709	714	720	725	730	736	
808	741	747	752	757	763	768	773	779	784	789	
809	795	800	806	811	816	822	827	832	838	843	
810	849	854	859	865	870	875	881	886	891	897	
811	902	907	913	918	924	929	934	940	945	950	**6**
812	956	961	966	972	977	982	988	993	998	*004	1 | 0.6
813	91 009	014	020	025	030	036	041	046	052	057	2 | 1.2
814	062	068	073	078	084	089	094	100	105	110	3 | 1.8
815	116	121	126	132	137	142	148	153	158	164	4 | 2.4
816	169	174	180	185	190	196	201	206	212	217	5 | 3.0
817	222	228	233	238	243	249	254	259	265	270	6 | 3.6
818	275	281	286	291	297	302	307	312	318	323	7 | 4.2
819	328	334	339	344	350	355	360	365	371	376	8 | 4.8
820	381	387	392	397	403	408	413	418	424	429	9 | 5.4
821	434	440	445	450	455	461	466	471	477	482	
822	487	492	498	503	508	514	519	524	529	535	
823	540	545	551	556	561	566	572	577	582	587	
824	593	598	603	609	614	619	624	630	635	640	
825	645	651	656	661	666	672	677	682	687	693	
826	698	703	709	714	719	724	730	735	740	745	
827	751	756	761	766	772	777	782	787	793	798	
828	803	808	814	819	824	829	834	840	845	850	
829	855	861	866	871	876	882	887	892	897	903	
830	908	913	918	924	929	934	939	944	950	955	
831	960	965	971	976	981	986	991	997	*002	*007	**5**
832	92 012	018	023	028	033	038	044	049	054	059	1 | 0.5
833	065	070	075	080	085	091	096	101	106	111	2 | 1.0
834	117	122	127	132	137	143	148	153	158	163	3 | 1.5
835	169	174	179	184	189	195	200	205	210	215	4 | 2.0
836	221	226	231	236	241	247	252	257	262	267	5 | 2.5
837	273	278	283	288	293	298	304	309	314	319	6 | 3.0
838	324	330	335	340	345	350	355	361	366	371	7 | 3.5
839	376	381	387	392	397	402	407	412	418	423	8 | 4.0
840	428	433	438	443	449	454	459	464	469	474	9 | 4.5
841	480	485	490	495	500	505	511	516	521	526	
842	531	536	542	547	552	557	562	567	572	578	
843	583	588	593	598	603	609	614	619	624	629	
844	634	639	645	650	655	660	665	670	675	681	
845	686	691	696	701	706	711	716	722	727	732	
846	737	742	747	752	758	763	768	773	778	783	
847	788	793	799	804	809	814	819	824	829	334	
848	840	845	850	855	860	865	870	875	881	886	
849	891	896	901	906	911	916	921	927	932	937	
850	942	947	952	957	962	967	973	978	983	988	
N.	L. 0	1	2	3	4	5	6	7	8	9	P. P.

TABLE—(*Continued*).

N.	L. 0	1	2	3	4	5	6	7	8	9
850	92 942	947	952	957	962	967	973	978	983	988
851	993	998	*003	*008	*013	*018	*024	*029	*034	*039
852	93 044	049	054	059	064	069	075	080	085	090
853	095	100	105	110	115	120	125	131	136	141
854	146	151	156	161	166	171	176	181	186	192
855	197	202	207	212	217	222	227	232	237	242
856	247	252	258	263	268	273	278	283	288	293
857	298	303	308	313	318	323	328	334	339	344
858	349	354	359	364	369	374	379	384	389	394
859	399	404	409	414	420	425	430	435	440	445
860	450	455	460	465	470	475	480	485	490	495
861	500	505	510	515	520	526	531	536	541	546
862	551	556	561	566	571	576	581	586	591	596
863	601	606	611	616	621	626	631	636	641	646
864	651	656	661	666	671	676	682	687	892	897
865	702	707	712	717	722	727	732	737	742	747
866	752	757	762	767	772	777	782	787	792	797
867	802	807	812	817	822	827	832	837	842	847
868	852	857	862	867	872	877	882	887	892	897
869	902	907	912	917	922	927	932	937	942	947
870	952	957	962	967	972	977	982	987	992	997
871	94 002	007	012	017	022	027	032	037	042	047
872	052	057	062	067	072	077	082	086	091	096
873	101	106	111	116	121	126	131	136	141	146
874	151	156	161	166	171	176	181	186	191	196
875	201	206	211	216	221	226	231	236	240	245
876	250	255	260	265	270	275	280	285	290	295
877	300	305	310	315	320	325	330	335	340	345
878	349	354	359	364	369	374	379	384	389	394
879	399	404	409	414	419	424	429	433	438	443
880	448	453	458	463	468	473	478	483	488	493
881	498	503	507	512	517	522	527	532	537	542
882	547	552	557	562	567	571	576	581	586	591
883	596	601	606	611	616	621	626	630	635	640
884	645	650	655	660	665	670	675	680	685	689
885	694	699	704	709	714	719	724	729	734	738
886	743	748	753	758	763	768	773	778	783	787
887	792	797	802	807	812	817	822	827	832	836
888	841	846	851	856	861	866	871	876	880	885
889	890	895	900	905	910	915	919	924	929	934
890	939	944	949	954	959	963	968	973	978	983
891	988	993	998	*002	*007	*012	*017	*022	*027	*032
892	95 036	041	046	051	056	061	066	071	075	080
893	085	090	095	100	105	109	114	119	124	129
894	134	139	143	148	153	158	163	168	173	177
895	182	187	192	197	202	207	211	216	221	226
896	231	236	240	245	250	255	260	265	270	274
897	279	284	289	294	299	303	308	313	318	323
898	328	332	337	342	347	352	357	361	366	371
899	376	381	386	390	395	400	405	410	415	419
900	424	429	434	439	444	448	453	458	463	468
N.	L. 0	1	2	3	4	5	6	7	8	9

P. P.

	6
1	0.6
2	1.2
3	1.8
4	2.4
5	3.0
6	3.6
7	4.2
8	4.8
9	5.4

	5
1	0.5
2	1.0
3	1.5
4	2.0
5	2.5
6	3.0
7	3.5
8	4.0
9	4.5

	4
1	0.4
2	0.8
3	1.2
4	1.6
5	2.0
6	2.4
7	2.8
8	3.2
9	3.6

P. P.

TABLE—(*Continued*).

N.	L. 0	1	2	3	4	5	6	7	8	9
900	95 424	429	434	439	444	448	453	458	463	468
901	472	477	482	487	492	497	501	506	511	516
902	521	525	530	535	540	545	550	554	559	564
903	569	574	578	583	588	593	598	602	607	612
904	617	622	626	631	636	641	646	650	655	660
905	665	670	674	679	684	689	694	698	703	708
906	713	718	722	727	732	737	742	746	751	756
907	761	766	770	775	780	785	789	794	799	804
908	809	813	818	823	828	832	837	842	847	852
909	856	861	866	871	875	880	885	890	895	899
910	904	909	914	918	923	928	933	938	942	947
911	952	957	961	966	971	976	980	985	990	995
912	999	*004	*009	*014	*019	*023	*028	*033	*038	*042
913	96 047	052	057	061	066	071	076	080	085	090
914	095	099	104	109	114	118	123	128	133	137
915	142	147	152	156	161	166	171	175	180	185
916	190	194	199	204	209	213	218	223	227	232
917	237	242	246	251	256	261	265	270	275	280
918	284	289	294	298	303	308	313	317	322	327
919	332	336	341	346	350	355	360	365	369	374
920	379	384	388	393	398	402	407	412	417	421
921	426	431	435	440	445	450	454	459	464	468
922	473	478	483	487	492	497	501	506	511	515
923	520	525	530	534	539	544	548	553	558	562
924	567	572	577	581	586	591	595	600	605	609
925	614	619	624	628	633	638	642	647	652	656
926	661	666	670	675	680	685	689	694	699	703
927	708	713	717	722	727	731	736	741	745	750
928	755	759	764	769	774	778	783	788	792	797
929	802	806	811	816	820	825	830	834	839	844
930	848	853	858	862	867	872	876	881	886	890
931	895	900	904	909	914	918	923	928	932	937
932	942	946	951	956	960	965	970	974	979	984
933	988	993	997	*002	*007	*011	*016	*021	*025	*030
934	97 035	039	044	049	053	058	063	067	072	077
935	081	086	090	095	100	104	109	114	118	123
936	128	132	137	142	146	151	155	160	165	169
937	174	179	183	188	192	197	202	206	211	216
938	220	225	230	234	239	243	248	253	257	262
939	267	271	276	280	285	290	294	299	304	308
940	313	317	322	327	331	336	340	345	350	354
941	359	364	368	373	377	382	387	391	396	400
942	405	410	414	419	424	428	433	437	442	447
943	451	456	460	465	470	474	479	483	488	493
944	497	502	506	511	516	520	525	529	534	539
945	543	548	552	557	562	566	571	575	580	585
946	589	594	598	603	607	612	617	621	626	630
947	635	640	644	649	653	658	663	667	672	676
948	681	685	690	695	699	704	708	713	717	722
949	727	731	736	740	745	749	754	759	763	768
950	772	777	782	786	791	795	800	804	809	813
N.	L. 0	1	2	3	4	5	6	7	8	9

P. P.

	5
1	0.5
2	1.0
3	1.5
4	2.0
5	2.5
6	3.0
7	3.5
8	4.0
9	4.5

	4
1	0.4
2	0.8
3	1.2
4	1.6
5	2.0
6	2.4
7	2.8
8	3.2
9	3.6

P. P.

TABLE—(*Continued*).

N.	L. 0	1	2	3	4	5	6	7	8	9
950	97 772	777	782	786	791	795	800	804	809	813
951	818	823	827	832	836	841	845	850	855	859
952	864	868	873	877	882	886	891	896	900	905
953	909	914	918	923	928	932	937	941	946	950
954	955	959	964	968	973	978	982	987	991	996
955	98 000	005	009	014	019	023	028	032	037	041
956	046	050	055	059	064	068	073	078	082	087
957	091	096	100	105	109	114	118	123	127	132
958	137	141	146	150	155	159	164	168	173	177
959	182	186	191	195	200	204	209	214	218	223
960	227	232	236	241	245	250	254	259	263	268
961	272	277	281	286	290	295	299	304	308	313
962	318	322	327	331	336	340	345	349	354	358
963	363	367	372	376	381	385	390	394	399	403
964	408	412	417	421	426	430	435	439	444	448
965	453	457	462	466	471	475	480	484	489	493
966	498	502	507	511	516	520	525	529	534	538
967	543	547	552	556	561	565	570	574	579	583
968	588	592	597	601	605	610	614	619	623	628
969	632	637	641	646	650	655	659	664	668	673
970	677	682	686	691	695	700	704	709	713	717
971	722	726	731	735	740	744	749	753	758	762
972	767	771	776	780	784	789	793	798	802	807
973	811	816	820	825	829	834	838	843	847	851
974	856	860	865	869	874	878	883	887	892	896
975	900	905	909	914	918	923	927	932	936	941
976	945	949	954	958	963	967	972	976	981	985
977	989	994	998	*003	*007	*012	*016	*021	*025	*029
978	99 034	038	043	047	052	056	061	065	069	074
979	078	083	087	092	096	100	105	109	114	118
980	123	127	131	136	140	145	149	154	158	162
981	167	171	176	180	185	189	193	198	202	207
982	211	216	220	224	229	233	238	242	247	251
983	255	260	264	269	273	277	282	286	291	295
984	300	304	308	313	317	322	326	330	335	339
985	344	348	352	357	361	366	370	374	379	383
986	388	392	396	401	405	410	414	419	423	427
987	432	436	441	445	449	454	458	463	467	471
988	476	480	484	489	493	498	502	506	511	515
989	520	524	528	533	537	542	546	550	555	559
990	564	568	572	577	581	585	590	594	599	603
991	607	612	616	621	625	629	634	638	642	647
992	651	656	660	664	669	673	677	682	686	691
993	695	699	704	708	712	717	721	726	730	734
994	739	743	747	752	756	760	765	769	774	778
995	782	787	791	795	800	804	808	813	817	822
996	826	830	835	839	843	848	852	856	861	865
997	870	874	878	883	887	891	896	900	904	909
998	913	917	922	926	930	935	939	944	948	952
999	957	961	965	970	974	978	983	987	991	996
1000	00 000	004	009	013	017	022	026	030	035	039
N.	L. 0	1	2	3	4	5	6	7	8	9

P. P.

	5
1	0.5
2	1.0
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4	2.0
5	2.5
6	3.0
7	3.5
8	4.0
9	4.5

	4
1	0.4
2	0.8
3	1.2
4	1.6
5	2.0
6	2.4
7	2.8
8	3.2
9	3.6

P. P.

GEOMETRY

1. The sum of all the angles formed on one side of a straight line equals two right angles, or 180°.

2. The sum of all the angles formed around a point equals four right angles, or 360°.

3. When two straight lines intersect each other, the opposite or vertical angles are equal.

4. If two angles have their sides parallel, they are equal.

5. If two triangles have two sides and the included angle of the one equal to two sides and the included angle of the other, they are equal in all their parts.

6. If two triangles have two angles and the included side of the one equal to two angles and the included side of the other, they are equal in all their parts.

7. In any triangle, the greater side is opposite the greater angle, and the greater angle is opposite the greater side.

8. The sum of the lengths of any two sides of a triangle is greater than the length of the third side.

9. In an isosceles triangle, the angles opposite the equal sides are equal.

10. In any triangle, the sum of the three angles is equal to two right angles, or 180°.

11. If two angles of a triangle are given, the third may be found by subtracting their sum from two right angles, or 180°.

12. A triangle must have at least two acute angles, and can have but one obtuse or one right angle.

13. In any triangle, a perpendicular let fall from the apex to the base is shorter than either of the two other sides.

14. In any parallelogram, the opposite sides and angles are equal each to each.

15. The diagonals divide any parallelogram into two equal triangles.

16. The diagonals of a parallelogram bisect each other; that is, they divide each other into equal parts.

17. If the sides of a polygon are produced in the same direction, the sum of the exterior angles will equal four right angles.

18. The sum of the interior angles of a polygon is equal to twice as many right angles as the polygon has sides, less

four right angles. For example, the sum of the interior angles of a quadrilateral is $(2\times4)-4=4$ right angles; the sum of the interior angles of a pentagon is $(2\times5)-4=6$ right angles; the sum of the interior angles of a hexagon is $(2\times6)-4=8$ right angles.

19. In equiangular polygons, each interior angle equals the sum divided by the number of sides.

20. The square described on the hypotenuse of a right-angled triangle is equal to the sum of the squares described on the other two sides. Thus, in a right-angled triangle whose base is 20 ft. and altitude 10 ft., the square of the hypotenuse equals the square of 20 + the square of 10, or 500; then the hypotenuse equals the square root of 500, or 22.3607 ft.

21. Having the hypotenuse and one side of a right-angled triangle, the other side may be found by subtracting from the square of the hypotenuse the square of the other known side; the remainder will be the square of the required side.

22. Triangles that have an angle in each equal, are to each other as the product of the sides including those equal angles.

23. Similar triangles are to each other as the squares of their corresponding sides.

24. The perimeters of similar polygons are to each other as any two corresponding sides, and their areas are to each other as the squares of those sides.

25. The diameter of a circle is greater than any chord.

26. Any radius that is perpendicular to a chord, bisects the chord and the arc subtended by the chord.

27. Through three points not in the same line, a circumference may be made to pass. For example, draw two lines connecting the three points and erect perpendiculars from the centers of each of these two lines; the point of intersection of the perpendiculars will be the center of the circle.

28. The circumferences of circles are to each other as their radii, and their areas are to each other as the squares of their radii.

EXAMPLE 1.—If the circumference of a circle is 62.83 in. and its radius is 10 in., what is the circumference of a circle whose radius is 15 in.?

SOLUTION.—Applying the principle just given, the circumference is

$$10 : 15 = 62.83 : 94.245 \text{ in.}$$

EXAMPLE 2.—If a circle 6 in. in diameter has an area of 28.274 sq. in., what is the area of a circle 12 in. in diameter?

SOLUTION.—Applying the principle just given, the area is

$$3^2 : 6^2 = 28.274 : 113.096 \text{ sq. in.}$$

MENSURATION

In the following formulas, the letters have the meanings here given, unless otherwise stated.

D = larger diameter;
d = smaller diameter;
R = radius corresponding to D;
r = radius corresponding to d;
p = perimeter of circumference;
C = area of convex surface, that is the area of flat surface that can be rolled into the shape shown;
S = area of entire surface ends $= C +$ area of the end or ends;
A = area of plane figure;
$\pi = 3.1416$, nearly, that is the ratio of any circumference to its diameter;
V = volume of solid.

The other letters used will be found on the cuts.

CIRCLE

$$p = \pi d = 3.1416d$$

$$p = 2\pi r = 6.2832r$$

$$p = 2\sqrt{\pi A} = 3.5449\sqrt{A}$$

$$p = \frac{2A}{r} = \frac{4A}{d}$$

$$d = \frac{p}{\pi} = \frac{p}{3.1416} = .3183p$$

$$d = 2\sqrt{\frac{A}{\pi}} = 1.1284\sqrt{A}$$

$$r = \frac{p}{2\pi} = \frac{p}{6.2832} = .1592p$$

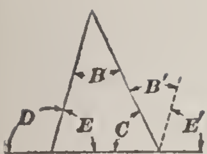
$$r=\sqrt{\frac{A}{\pi}}=.5642\sqrt{A}$$

$$A=\frac{\pi d^2}{4}=.7854d^2$$

$$A=\pi r^2=3.1416r^2$$

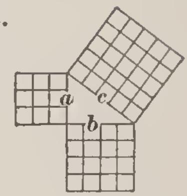
$$A=\frac{pr}{2}=\frac{pd}{4}$$

TRIANGLES



$D = B+C$	$B+B+C=180°$
$B = D-C$	$E'+B+C=180°$
$E' = E$	$B'=B$

The above letters refer to angles.

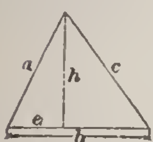


For a right-angled triangle, c being the hypotenuse,

$$c=\sqrt{a^2+b^2}$$

$$a=\sqrt{c^2-b^2}$$

$$b=\sqrt{c^2-a^2}$$

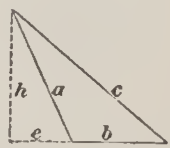


c = length of side opposite an acute angle of an oblique-angled triangle.

$$c=\sqrt{a^2+b^2-2be}$$

$$h=\sqrt{a^2-e^2}$$

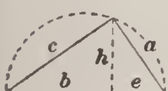
c = length of side opposite an obtuse angle of an oblique-angled triangle.



$$c=\sqrt{a^2+b^2+2be}$$

$$h=\sqrt{a^2-e^2}$$

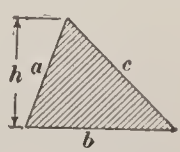
For a triangle inscribed in a semicircle; i. e., any right-angled triangle, $c : b=a : h$



$$h=\frac{ab}{c}=\frac{ce}{a}$$

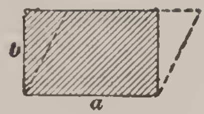
$$a : b+e=e : a=h : c$$

For any triangle,



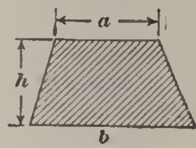
$$A=\frac{bh}{2}=\tfrac{1}{2}bh$$

$$A=\frac{b}{2}\sqrt{a^2-\left(\frac{a^2+b^2-c^2}{2b}\right)^2}$$



RECTANGLE AND PARALLELOGRAM

$$A = ab$$

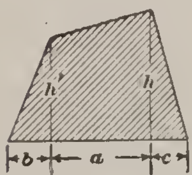


TRAPEZOID

$$A = \tfrac{1}{2}h(a+b)$$

TRAPEZIUM

Divide into two triangles and a trapezoid.



$$A = \tfrac{1}{2}bh' + \tfrac{1}{2}a(h'+h) + \tfrac{1}{2}ch$$

or, $A = \tfrac{1}{2}[bh' + ch + a(h'+h)]$

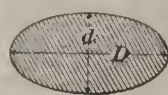
Or, divide into two triangles by drawing a diagonal. Consider the diagonal as the base of both triangles, call its length l; call the altitudes of the triangles h_1 and h_2; then

$$A = \tfrac{1}{2}l(h_1 + h_2)$$

ELLIPSE

$$p^* = \pi\sqrt{\frac{D^2+d^2}{2} - \frac{(D-d)^2}{8.8}}$$

$$A = \frac{\pi}{4}Dd = .7854Dd$$



SECTOR

$$A = \tfrac{1}{2}lr$$

$$A = \frac{\pi r^2 E}{360} = .008727r^2E$$

l = length of arc

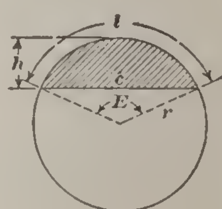
SEGMENT

$$A = \tfrac{1}{2}\,[lr - c\,(r-h)]$$

$$A = \frac{\pi r^2 E}{360} - \frac{c}{2}(r-h)$$

$$l = \frac{\pi r E}{180} = .0175rE$$

$$E = \frac{180l}{\pi r} = 57.2956\,\frac{l}{r}$$



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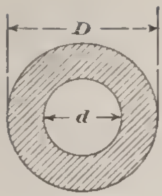
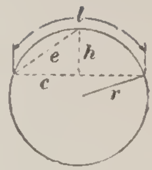
*The perimeter of an ellipse cannot be exactly determined without a very elaborate calculation, and this formula is merely an approximation giving fairly close results.



RING

$$A=\frac{\pi}{4}(D^2-d^2)$$

CHORD

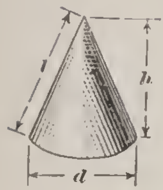


c = length of chord

$$r=\frac{c^2+4h^2}{8h}=\frac{e^2}{2h}$$

$$c=2\sqrt{2hr-h^2}$$

$$l=\frac{8e-c}{3}\text{, approximately}$$



CONE

$$C=\tfrac{1}{2}\pi dl=\pi rl$$

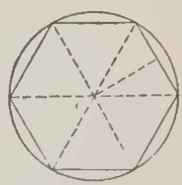
$$S=\pi rl+\pi r^2=\pi r\sqrt{r^2+h^2}+\pi r^2$$

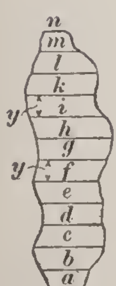
$$V=\frac{\pi d^2}{4}\times\frac{h}{3}=\frac{.7854d^2h}{3}=\frac{p^2h}{12\pi}$$

REGULAR POLYGONS

Divide the polygon into equal triangles and find the sum of the partial areas. Otherwise, square the length of one side and multiply by proper number from the following table:

Name	*No. Sides*	*Multiplier*
Triangle.	3	.433
Square.	4	1.000
Pentagon	5	1.720
Hexagon.	6	2.598
Heptagon	7	3.634
Octagon.	8	4.828
Nonagon.	9	6.182
Decagon	10	7.694





IRREGULAR AREAS

Divide the area into trapezoids, triangles, parts of circles, etc., and find the sum of the partial areas. If the figure is very irregular, the approximate area may be found as follows: Divide the figure into trapezoids by equidistant parallel lines b, c, d, etc. The lengths

of these lines being measured, then, calling a the first and n the last length, and y the width of strips,

$$A = y\left(\frac{a+n}{2} + b + c + \text{etc.} + m\right)$$

PLANE TRIGONOMETRY

DEFINITIONS

Plane trigonometry treats of the solution of plane triangles. Every triangle consists of six parts, three sides and three angles. These parts are so related that when three are given, one being a side, the other parts may be found.

An *angle* is measured by the arc included between its sides, the center of the circumference being at the vertex of the angle.

For measuring angles, the circumference is divided into 360 equal parts, called *degrees*; each degree is divided into 60 equal parts called *minutes*.

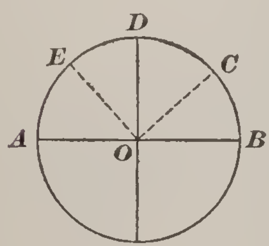


FIG. 1

A *quadrant* is one-fourth the circumference of a circle, or 90°.

The *complement* of an arc is 90° minus the arc; in Fig. 1, DC is the complement of BC, and the angle DOC is the complement of BOC.

The *supplement* of an arc is 180° minus the arc; in Fig. 1, AE is the supplement of the arc BDE, and the angle AOE is the supplement of the angle BOE.

In trigonometry, instead of comparing the angles of triangles or the arcs that measure them, the trigonometric functions known as the sine, cosine, tangent, cotangent, secant, and cosecant are compared.

The *sine*, or sin, of an arc is the perpendicular let fall from one extremity of the arc on the diameter that passes through the other extremity; in Fig. 2, CD is the sine of the arc AC.

The *cosine*, or cos, of an arc is the sine of its complement; or it is the distance from the foot of the sine to the center of the circle; in Fig. 2, CE or OD equals the cosine of arc AC.

The *tangent*, or tan, of an arc is a line perpendicular to the radius at one extremity of an arc and limited by a line passing through the center of the circle and the other extremity; in Fig. 2, AT is the tangent of AC.

The *cotangent*, or cot, of an arc is equal to the tangent of the complement of the arc; in Fig. 2, BT' is the cotangent of AC.

The *secant*, or sec, of an arc is a line drawn from the center of the circle through one extremity of the arc, and limited by a tangent at the other extremity; in Fig. 2, OT is the secant of AC.

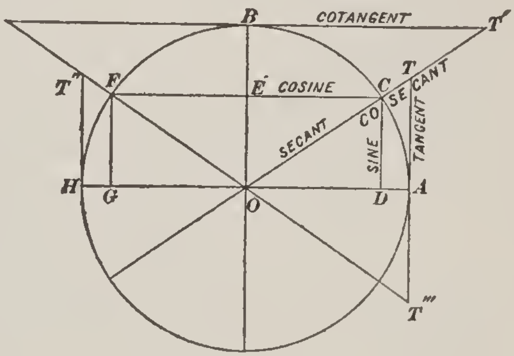


FIG. 2

The *cosecant*, or cosec, of an arc is the secant of the complement of the arc; in Fig. 2, the line OT' is the cosecant of the arc AC.

The *versed sine* of an arc is that part of the diameter included between the extremity of the arc and the foot of the sine; in Fig. 2, DA is the versed sine of AC.

The *coversed sine* is the versed sine of the complement of the arc; in Fig. 2, BE is the coversed sine of AC.

From the above definitions, we derive the following simple principles:

1. The sine of an arc equals the sine of its supplement, and the cosine of an arc equals the cosine of its supplement.

2. The tangent of an arc equals the tangent of its supplement, and the cotangent of an arc equals the cotangent of its supplement.

3. The secant of an arc equals the secant of its supplement, and the cosecant equals the cosecant of its supplement. Thus,

$\sin\ 70° = \sin\ 110°$	$\cos\ 70° = \cos\ 110°$
$\tan\ 70° = \tan\ 110°$	$\cot\ 70° = \cot\ 110°$
$\sec\ 70° = \sec\ 110°$	$\text{cosec}\ 70° = \text{cosec}\ 110°$

Thus, to find the sin $120°\ 30'$, look for the sin $180 - 120°\ 30'$, or $59°\ 30'$, etc.

In the right-angled triangle, *xyz*, Fig. 3, the following relations hold:

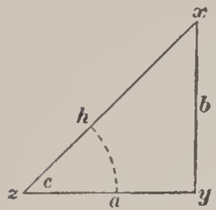


FIG. 3

$$\sin c = \frac{b}{h} \qquad \cos c = \frac{a}{h}$$

$$\tan c = \frac{b}{a} \qquad \cot c = \frac{a}{b}$$

$$\sec c = \frac{h}{a} \qquad \operatorname{cosec} c = \frac{h}{b}$$

The functions of the sum and difference of two angles are:

$$\sin (A+B) = \sin A \cos B + \cos A \sin B$$
$$\cos (A+B) = \cos A \cos B - \sin A \sin B$$
$$\sin (A-B) = \sin A \cos B - \cos A \sin B$$
$$\cos (A-B) = \cos A \cos B + \sin A \sin B$$

Natural sines, tangents, etc. are calculated for a circle having a radius of unity, and logarithmic sines, tangents, etc. are calculated for a circle whose radius is 10,000,000,000.

EXAMPLES IN SOLUTION OF TRIANGLES

To Determine Height of Vertical Object Standing on Horizontal Plane.—Measure from the foot of the object any convenient horizontal distance *A B*, Fig. 1; at the point *A*, take the angle of elevation *B A C*. Then, as *B* is known to be a right angle, two angles and the included side of a triangle are known. Assuming that the line *A B* is 300 ft. and the angle $B A C = 40°$, the angle $C = 180° - (90° + 40°) = 50°$. Then, $\sin C : A B = \sin A : BC$, or, .766044 : 300 = .642788 : ( ) or, 251.73 +ft. Or, by logarithms:

$$\begin{array}{rr}
\text{Log } 300 = & 2.477121 \\
\text{Log } \sin 40° = & \underline{9.808067} \\
 & 12.285188 \\
\text{Log } \sin 50° = & \underline{9.884254} \\
 & 2.400934
\end{array}$$
or log of 251.73+ft.

Hence, $B C = 251.73+$ ft.

To Find Distance of Vertical Object Whose Height is Known. At a point *A*, Fig. 2, take the angle of elevation to the top of the object. Knowing that the angle *B* is a right angle, the angles *B* and *A* and the side *B C* of a triangle are known. Assuming

that the side $BC = 200$ ft. and the angle $A = 30°$, the triangle is: Angle $A = 30°$, $B = 90°$, $C = 60°$, and the side $BC = 200$ ft.

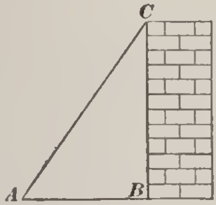


FIG. 1

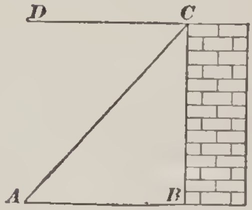


FIG. 2

Then, $\sin A : BC = \sin C : AB$, or $.5 : 200 = .866025 :$, or 346.41 ft. By logarithms:

$$
\begin{array}{rr}
\text{Log } 200 = & 2.301030 \\
\text{Log } \sin 60° = & \underline{9.937531} \\
 & 12.238561 \\
\text{Log } \sin 30° = & \underline{9.698970} \\
 & 2.539591 \text{ or log of } 346.41 \text{ ft.}
\end{array}
$$

To Find Distance of Inaccessible Object.—Measure a horizontal base line AB, Fig. 3, and take the angles formed by the lines BAC and ABC; this gives the two angles and the included side. Assuming the angle $A = 60°$, the angle $B = 50°$. and the side $AB = 500$ ft., angle $C = 180° - (60° + 50°) = 70°$. Then,

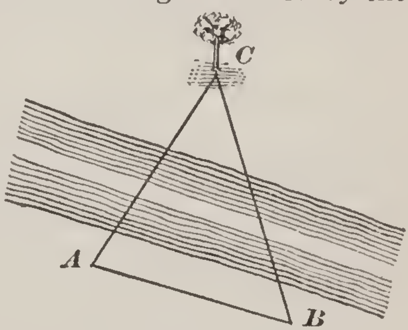


FIG. 3

$\sin 70° : AB = \sin A : BC$,

and

$\sin 70° : AB = \sin B : AC$;

or, $.939693 : 500 = .866025 : BC$, or 460.8+,

and $.939693 : 500 = .766044 : AC$, or 407.6+.

By logarithms:

$$
\begin{array}{rr}
\text{Log } 500 = & 2.698970 \\
\text{Log } \sin 60° = & \underline{9.937531} \\
 & 12.636501 \\
\text{Log } \sin 70° = & \underline{9.972986} \\
 & 2.663515 = \text{log of } 460.8+
\end{array}
$$

$$\begin{array}{rr} \text{Log } 500 = & 2.698970 \\ \text{Log sin } 50° = & 9.842254 \\ \hline & 12.583224 \\ \text{Log sin } 70° = & 9.972986 \\ \hline & 2.610238 = \text{log of } 407.6+ \end{array}$$

To Find Distance Between Two Objects Separated by an Impassable Barrier.—Select any convenient station, as C, Fig. 4, measure the lines CA and CB, and the angle included between these sides; this gives the two sides and the included angle. Assuming angle $C = 60°$, the side $CA = 600$ ft., and side $CB = 500$ ft.,

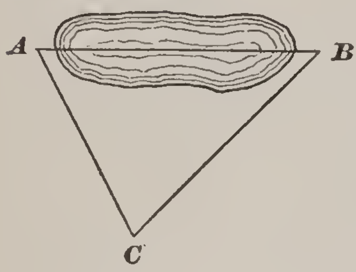


FIG. 4

$$CA + CB : CA - CB = \tan \frac{A+B}{2} : \tan \frac{B-A}{2}$$

Then, $$\frac{A+B}{2} = \frac{180° - 60°}{2}, \text{ or } 60°$$

Then, $$1{,}100 : 100 = \tan 60° : \tan \frac{B-A}{2};$$

or, $1{,}100 : 100 = 1.732050 : .157459$, or tangent of $\frac{B-A}{2}$, or $8° 57'$.

Then, $60° + 8° 57' = 68° 57'$, or angle B, and $60° - 8° 57' = 51° 03'$, or angle A. Having found the angles, find the third side by the method given in connection with Fig. 1.

The foregoing formula, worked out by logarithms, is as follows:

$$\begin{array}{rr} \text{Log } 100 = & 2.000000 \\ \text{Log Tan } 60° = & 10.238561 \\ \hline & 12.238561 \\ \text{Log } 1{,}100 = & 3.041393 \\ \hline & 9.197168 = \text{log tan of } \frac{B-A}{2}, \text{ or } 8° 57' \end{array}$$

Then, $60° + 8° 57' = 68° 57'$, or angle B, and $60° - 8° 57' = 51° 03'$, or angle A.

NOTE.—The greater angle is always opposite the greater side.

To Find Height of Vertical Object Standing Upon Inclined Plane.—Measure any convenient distance $D\ C$, Fig. 5, on a line from the foot of the object, and, at the point D, measure the angles of elevation $E\ D\ A$ and $E\ D\ B$ to foot and top of tower. This gives two triangles, both of which may be solved by the method given in connection with Fig. 1, and the height above D of both the foot and top will be known. The difference between them is the height of the tower.

To Find Height of Inaccessible Object Above a Horizontal Plane.—*First Method.*—Measure any convenient horizontal line $A\ B$, Fig. 6, directly toward the object, and take the angles

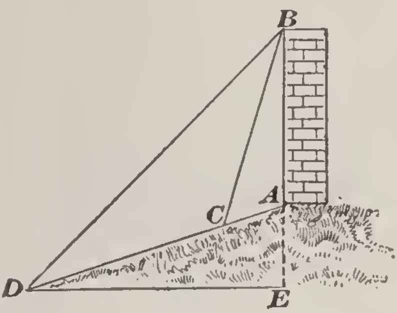


FIG. 5

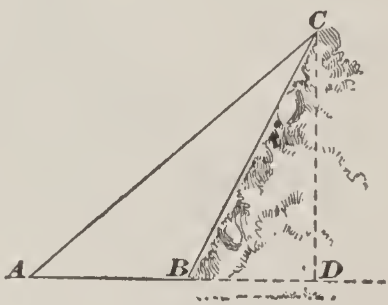


FIG. 6

of elevation at A and B. Assuming the line $A\ B = 1{,}200$ ft., the angle $A = 25°$, and the angle $D B C = 40°$; then angle $A B C = 180° - 40° = 140°$. Then, having the side $B\ C$, and the angle $D\ B\ C = 40°$, and the angle $B\ D\ C = 90°$, we find the side $C\ D$ by the same method given in connection with Fig. 1.

Second Method.—If it is not convenient to measure a horizontal base line toward the object, measure any line $A\ B$, Fig. 7, and also measure the horizontal angles $B\ A\ D$, $A\ B\ D$, and the angle of elevation $D\ B\ C$. Then, by means of the two triangles $A\ B\ D$ and $C\ B\ D$, the height $C\ D$ can be found. With the line $A\ B$ and the angles $B\ A\ D$ and $A\ B\ D$ known, two angles and the included side are known. The third angle is then readily found and the side $B\ D$ can be found. In the triangle $B D C$ the angle B is known; by measurement, $D = 90°$, and the side $B\ D$ is known. Then, the side $C\ D$, or the vertical height, can be found by the method given in connection with Fig. 1.

To Find Distance Between Two Inaccessible Objects When Points Can Be Found From Which Both Objects Can Be Seen. Wishing to know the horizontal distance between a tree and a house on the opposite side of a river, measure the line $A\ B$, Fig. 8, and, at point A, take the angles $D\ A\ C$, and $D\ A\ B$, and, at the point B, take the angles $C\ B\ A$ and $C\ B\ D$. Assume the length of $A\ B = 400$ ft.; angle $D\ A\ C = 56°\ 30'$; angle $D\ A\ B = 42°\ 24'$; angle $C\ B\ A = 44°\ 36'$; angle $C\ B\ D = 68°\ 50'$. In the triangle $A\ B\ D$, $A\ B = 400$ ft., angle $D\ A\ B = 42°\ 24'$, angle $A\ B\ D = (44°\ 36' + 68°\ 50') = 113°\ 26'$, and angle $A\ D\ B = 180° - (42°\ 24' + 113°\ 26') = 24°\ 10'$.

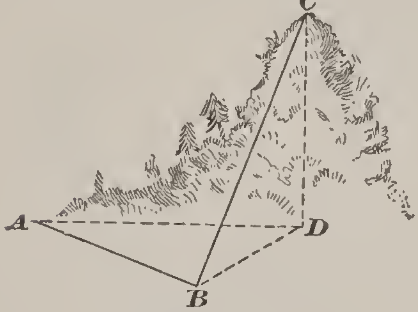


FIG. 7

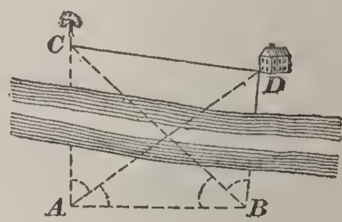


FIG. 8

Then, according to the method given in connection with Fig. 1, find the side $D\ B$; this gives three angles and two sides of the triangle $A\ D\ B$. The third side $A\ D$ is found by the same method. In the triangle $A\ B\ C$ the angles $A\ B\ C$ and $B\ A\ C$, and the distance $A\ B$ are known, so that the side $A\ C$ may be found. Then, in the triangle $A\ D\ C$, the sides $A\ D$ and $A\ C$, and the angle $D\ A\ C$, are known and the side $C\ D$ may be found by the method given in connection with Fig. 4.

TABLE OF TRIGONOMETRIC FUNCTIONS

The following table contains the natural sines, cosines, tangents, and cotangents of angles from 0° to 90°. Angles less than 45° are given in the first column and the names of the functions are given at the top; angles greater than 45° appear at the right-hand side of the page, and the names of the functions are given at the bottom. Thus, the second column contains

thc sincs of angles less than 45° and the cosines of angles greater than 45°; the sixth column contains the cotangents of angles less than 45° and the tangents of angles greater than 45°.

To find the function of an angle less than 45°, look in the first column for the angle, and at the top of the page for the name of the function; to find a function of an angle greater than 45°, look in the column at the right of the page for the angle and at the bottom of the page for the name of the function. The successive angles differ by an interval of 10′; they increase downwards in the left-hand column and upwards in the right-hand column. Thus, for angles less than 45° read down from top of page, and for angles greater than 45° read up from bottom of page.

The columns headed d contain the differences between the successive functions. For example, the sine of 32° 10′ is .5324 and the sine of 32° 20′ is .5348; the difference is $.5348 - .5324 = .0024$, so that 24 is written on the third column, just opposite the space between .5324 and .5348. In like manner the differences between the successive tabular values of the tangents are given in the fifth column, those between the cotangents in the seventh column, and those for the cosines in the ninth column. These differences in the functions correspond to a difference of 10′ in the angle; thus, when the angle 32° 10′ is increased by 10′, that is, to 32° 20′, the increase of the sine is .0024, or, as given in the table, 24. In the tabular difference, no attention is paid to the decimal point, as the difference is merely the number obtained by subtracting the last two or three figures of the smaller function from those of the larger.

These differences are used to obtain the sines, cosines, etc., of angles not given in the table. For example, suppose that the tangent of 27° 34′ is required. Looking in the table, it is found that the tangent of 27° 34′ is .5206, and (column 5) the difference for 10′ is 37, or when written in full .0037. As the difference for 1′ is $.0037 \div 10 = .00037$, the difference for 4′ is $.00037 \times 4 = .00148$. Adding this difference to the value of the tan 27° 30′ gives

$$\begin{aligned} \tan 27^\circ\, 30' &= .5206 \\ \text{difference for } 4' &= \underline{.00148} \\ \tan 27^\circ\, 34' &= .52208 \text{ or } .5221, \text{ to 4 places} \end{aligned}$$

Because only 4 decimal places are retained, the 8 in the fifth place is dropped and the fourth figure is increased by 1, as 8 is greater than 5.

Column of Proportional Parts.—To avoid multiplication, the column of proportional parts, headed P. P., is used. At the head of each table in this column is the difference for 10′, and below are the differences for any intermediate number of minutes from 1′ to 9′; these differences are written as whole numbers and decimal parts of same. In the example the difference for 10′ was 37; looking in the table with 37 at the head, the difference opposite 4 is 14.8; that opposite 7 is 25.9; and so on. For want of space, the differences for the cotangents for angles less than 45° (or the tangents of angles greater than 45°) have been omitted from the tables of proportional parts. The use of these functions should be avoided, if possible, for the differences change so rapidly that computation is likely to be inexact.

The method to be employed when dealing with these functions is shown in the following example, in which the tangent of 76° 34′ is required. Because this angle is greater than 45° it is found in the column at the right, which is to be read upwards. Opposite 76° 30′, in the sixth column, is found the number 4.1653; and corresponding to it, in the seventh column, is found the difference 540. As 540 is the difference for 10′, the difference for 4′ is $540 \times \frac{4}{10} = 216$. Putting this number in its true form and adding gives

$$\begin{aligned} \tan 76^\circ\, 30' &= 4.1653 \\ \text{difference for } 4' &= \underline{.0216} \\ \tan 76^\circ\, 34' &= 4.1869 \end{aligned}$$

Angles Containing Seconds.—When the angle contains a certain number of seconds, divide the number by 6, and take the whole number nearest to the quotient. Find this number in the table of proportional parts (under the proper difference), and take out the number that is opposite to it. Shift the decimal point one place to the left, and then add it to the partial function already found. The following examples represent the methods of using the tables of proportional parts for the different functions: For example, find the sine of 34° 26′ 44″.

$\sin 34° 20' = .5640$	Difference for $10' = 24$, or .0024
difference for $6' = .00144$	
difference for $44'' = .00017$	$\frac{44}{6} = 7\frac{1}{3}$. In the P.P. table, the number under 24 and opposite 7 is found to be 16.8. Shifting the decimal point one place to the left gives 1.68, or, 1.7, which when put in its decimal form is .00017.
$\sin 34° 26' 44'' = .5656$	

The tangent is found in the same way as the sine.

As the angle increases the value of the cosine decreases, therefore, to find the cosine of an angle, instead of adding the values corresponding to $6'$ and $44''$ to the function already found, subtract them from it. Thus, find the cosine of $34° 26' 44''$.

$\cos 34° 20' = .8258$	Difference for $10' = 17$, or .0017.
difference for $6' = .00102$	
difference for $44'' = .00012$	The number under the 17 and opposite the 7, in the P. P. table, is 11.9. Therefore take 1.19, or, say, 1.2, which may be written .00012.
total difference $= .00114$	
$.8247$	

Therefore, $\cos 34° 26' 44'' = .8258 - .0011 = .8247$. Only 4 decimal places are kept; therefore, the figure of the difference following the decimal point is dropped before subtracting.

The cotangent is found in the same manner.

To show the method when the angle is greater than 45°, suppose that it is required to find the sine of $68° 47' 22''$. In obtaining the difference, it must be remembered to choose the one between the sine of $68° 40'$ and the next angle above it, namely, $68° 50'$.

$\sin 68° 40' = .9315$	Difference for $10' = 10$, or .0010.
difference for $7' = .0007$	
difference for $19'' = .00004$	$\frac{22}{6} = 3\frac{2}{3}$, say 4. Under the 10 and opposite the 4 is the number 4.0; shifting the decimal point, gives .4, or .00004.
$\sin 68° 47' 22'' = .9322$	

As usual, only 4 decimal places are kept.

The tangent is found in a similar manner to the method just given.

As before, the cosine decreases as the angle increases; therefore, to find the cosine of 68° 47′ 22″, subtract the successive sine values corresponding to the increments in the angle.

cos 68° 40′ =	.3638	Difference for 10′ = 27, or, .0027.
difference for 7′ =	.00189	
difference for 22″ =	.00011	Under the 27 and opposite the 4 is the number 10.8; therefore, take 1.08 in this case, or, say, 1.1, which may be written .00011.
total difference =	.0020	
	.3618	

Therefore, cos 68° 47′ 22″ = .3638 − .002 = .3618.

The cotangent is found in the same way.

Applying Proportional Differences.—In finding the functions of an angle, the only difficulty likely to be encountered is to determine whether the difference obtained from the table of proportional parts is to be added or subtracted. This can be told by observing whether the function is increasing or decreasing as the angle increases. For example, take the angle 21°; its sine is .3584, and the following sines, reading downwards, are .3611, .3638, etc. Therefore, the sine of say 21° 6′ is greater than that of 21° and the difference for 6′ must be added. On the other hand, the cosine of 21° is .9336, and thc following cosines, reading downwards, are .9325, .9315, etc.; that is, as the angle grows larger the cosine decreases. The cosine of an angle between 21° and 21° 10′, say 21° 6′, must therefore lie between .9325 and .9315; that is, it must be smaller than .9325, which shows that the difference for 6′ must be subtracted from the cosine of 21°.

Finding the Angle.—Find the angle whose sine is .4943. Thc operation in this case may be arranged as follows:

	.4943	Difference for 10′ = 26, or .0026.
	.4924	= sin 29° 30′.
1st remainder	.0019	
	.00182	= difference for 7′.
2d remainder	.00008	

As .78 is the difference for .3′ or 18″, .4943 = sin. 29° 37′ 18″.

Looking down the second column, the sine next smaller than .4943 is found to be .4924, and the difference for 10′ to be 26.

The angle corresponding to .4924 is 29° 30′. Subtracting the .4924 from 4943, the first remainder is 19. Looking in the table of proportional parts, the part next lower than this difference is 18.2, opposite which is 7′. Subtracting this difference from the remainder gives .8, and, looking in the table, it is found that 7.8 with its decimal point moved one place to the left is nearest to the second difference. This is the difference for .3′ or 18″. Hence, the angle is 29° 30′ + 7′ + 18″ = 29° 37′ 18″.

Find the angle whose tangent is .8824.

	.8824	Difference for 10′ = 51, or .0051.
	.8796	= tan 41° 20′.
1st remainder	.0028	
	.00255	= difference for 5′.
2d remainder	.00025	

As 2.55 is the difference for .5′ or 30″, .8824 = tan 41° 25′ 30″.

In the examples just given, the minutes and seconds corresponding to the 1st and 2d remainders are added to the angle taken from the table. Thus, in the first example, an inspection of the table shows that the angle increases as the sine increases; hence, the angle whose sine is .4943 must be greater than 29° 30′, whose sine is .4924. For this reason the correction must be added to 29° 30′. The same reasoning applies to the second example.

Find the angle whose cosine is .7742.

	.7742	Difference for 10′ = 18, or .0018.
	.7735	= cos 39° 20′.
st remainder	.0007	
	.00054	= difference for 3′.
2d remainder	.00016	

As 1.62 is the difference for .9′ or 54″, 39° 20′ − 3′ 54″ = 39° 16′ 6″, which is the angle whose cosine is .7742.

Looking down the column, headed cos, the next smaller cosine is .7735, to which corresponds the angle 39° 20′. The difference for 10′ is 18. Subtracting, the remainder is 7, and the next lower number in the table of proportional parts is 5.4, which is the difference for 3′. Subtracting this from first remainder, the second remainder is 1.6, which is nearest 16.2 of table of proportional parts, if the decimal point of the latter

is moved to the left one place. As 16.2 corresponds to a difference of 9′, 1.62 corresponds to a difference of .9′, or 54″. Hence, the correction for the angle 39° 20′ is 3′ 54″. From the table, it appears that, as the cosine increases, the angle grows smaller; therefore, the angle whose cosine is .7742 must be smaller than the angle whose cosine is .7735, and the correction for the angle must be subtracted.

Find the angle whose cotangent is .9847.

$$\begin{array}{rll} & .9847 & \text{Difference for } 10' = 57, \text{ or } .0057. \\ & \underline{.9827} & = \cos 45^\circ\ 30'. \\ \text{1st remainder} & .0020 & \\ & \underline{.00171} & = \text{difference for } 3'. \\ \text{2d remainder} & .00029 & \end{array}$$

As 2.85 is the difference for .5′ or 30″, $45^\circ\ 30' - 3'\ 30'' = 45^\circ\ 26'\ 30''$, the angle whose cotangent is .9847.

In finding the angle corresponding to a function, as in the foregoing examples, the angles obtained may vary from the true angle by 2 or 3 sec.; in order to obtain the number of seconds accurately, the functions should contain 6 or 7 decimal places.

°	′	Sin.	d.	Tan.	d.	Cot.	d.	Cos.	d.		
0	0	0.0000		0.0000		infinit.		1.0000		0	90
	10	0.0029	29	0.0029	29	343.7737		1.0000	0	50	
	20	0.0058	29	0.0058	29	171.8854		1.0000	0	40	
	30	0.0087	29	0.0087	29	114.5887		1.0000	0	30	
	40	0.0116	29	0.0116	29	85.9398		0.9999	1	20	
	50	0.0145	29	0.0145	29	68.7501		0.9999	0	10	
1	0	0.0175	30	0.0175	30	57.2900		0.9998	1	0	89
	10	0.0204	29	0.0204	29	49.1039	81861	0.9998	0	50	
	20	0.0233	29	0.0233	29	42.9641	61398	0.9997	1	40	
	30	0.0262	29	0.0262	29	38.1885	47756	0.9997	0	30	
	40	0.0291	29	0.0291	29	34.3678	38207	0.9996	1	20	
	50	0.0320	29	0.0320	29	31.2416	31262	0.9995	1	10	
2	0	0.0349	29	0.0349	29	28.6363	26053	0.9994	1	0	88
	10	0.0378	29	0.0378	29	26.4316	22047	0.9993	1	50	
	20	0.0407	29	0.0407	29	24.5418	18898	0.9992	1	40	
	30	0.0436	29	0.0437	30	22.9038	16380	0.9990	2	30	
	40	0.0465	29	0.0466	29	21.4704	14334	0.9989	1	20	
	50	0.0494	29	0.0495	29	20.2056	12648	0.9988	1	10	
3	0	0.0523	29	0.0524	29	19.0811	11245	0.9986	2	0	87
	10	0.0552	29	0.0553	29	18.0750	10061	0.9985	1	50	
	20	0.0581	29	0.0582	29	17.1693	9057	0.9983	2	40	
	30	0.0610	29	0.0612	30	16.3499	8194	0.9981	2	30	
	40	0.0640	30	0.0641	29	15.6048	7451	0.9980	1	20	
	50	0.0669	29	0.0670	29	14.9244	6804	0.9978	2	10	
4	0	0.0698	29	0.0699	29	14.3007	6237	0.9976	2	0	86
	10	0.0727	29	0.0729	30	13.7267	5740	0.9974	2	50	
	20	0.0756	29	0.0758	29	13.1969	5298	0.9971	3	40	
	30	0.0785	29	0.0787	29	12.7062	4907	0.9969	2	30	
	40	0.0814	29	0.0816	29	12.2505	4557	0.9967	2	20	
	50	0.0843	29	0.0846	30	11.8262	4243	0.9964	3	10	
5	0	0.0872	29	0.0875	29	11.4301	3961	0.9962	2	0	85
	10	0.0901	29	0.0904	29	11.0594	3707	0.9959	3	50	
	20	0.0929	28	0.0934	30	10.7119	3475	0.9957	2	40	
	30	0.0958	29	0.0963	29	10.3854	3265	0.9954	3	30	
	40	0.0987	29	0.0992	29	10.0780	3074	0.9951	3	20	
	50	0.1016	29	0.1022	30	9.7882	2898	0.9948	3	10	
6	0	0.1045	29	0.1051	29	9.5144	2738	0.9945	3	0	84
	10	0.1074	29	0.1080	29	9.2553	2591	0.9942	3	50	
	20	0.1103	29	0.1110	30	9.0098	2455	0.9939	3	40	
	30	0.1132	29	0.1139	29	8.7769	2329	0.9936	3	30	
	40	0.1161	29	0.1169	30	8.5555	2214	0.9932	4	20	
	50	0.1190	29	0.1198	29	8.3450	2105	0.9929	3	10	
7	0	0.1219	29	0.1228	30	8.1443	2007	0.9925	4	0	83
	10	0.1248	29	0.1257	29	7.9530	1913	0.9922	3	50	
	20	0.1276	28	0.1287	30	7.7704	1826	0.9918	4	40	
	30	0.1305	29	0.1317	30	7.5958	1746	0.9914	4	30	
	40	0.1334	29	0.1346	29	7.4287	1671	0.9911	3	20	
	50	0.1363	29	0.1376	30	7.2687	1600	0.9907	4	10	
8	0	0.1392	29	0.1405	29	7.1154	1533	0.9903	4	0	82
	10	0.1421	29	0.1435	30	6.9682	1472	0.9899	4	50	
	20	0.1449	28	0.1465	30	6.8269	1413	0.9894	5	40	
	30	0.1478	29	0.1495	30	6.6912	1357	0.9890	4	30	
	40	0.1507	29	0.1524	29	6.5606	1306	0.9886	4	20	
	50	0.1536	29	0.1554	30	6.4348	1258	0.9881	5	10	
9	0	0.1564	28	0.1584	30	6.3138	1210	0.9877	4	0	81
		Cos.	d.	Cot.	d.	Tan.	d.	Sin.	d.	′	°

P. P.

	30	29	28	5	4
1	3.0	2.9	2.8	0.5	0.4
2	6.0	5.8	5.6	1.0	0.8
3	9.0	8.7	8.4	1.5	1.2
4	12.0	11.6	11.2	2.0	1.6
5	15.0	14.5	14.0	2.5	2.0
6	18.0	17.4	16.8	3.0	2.4
7	21.0	20.3	19.6	3.5	2.8
8	24.0	23.2	22.4	4.0	3.2
9	27.0	26.1	25.2	4.5	3.6

P. P.

°	′	Sin.	d.	Tan.	d.	Cot.	d.	Cos.	d.		
9	0	0.1564	29	0.1584	30	6.3138	1168	0.9877	5	0	81
	10	0.1593	29	0.1614	30	6.1970	1126	0.9872	4	50	
	20	0.1622	28	0.1644	29	6.0844	1086	0.9868	5	40	
	30	0.1650	29	0.1673	30	5.9758	1050	0.9863	5	30	
	40	0.1679	29	0.1703	30	5.8708	1014	0.9858	5	20	
	50	0.1708	28	0.1733	30	5.7694	981	0.9853	5	10	
10	0	0.1736	29	0.1763	30	5.6713	949	0.9848	5	0	80
	10	0.1765	29	0.1793	30	5.5764	919	0.9843	5	50	
	20	0.1794	28	0.1823	30	5.4845	890	0.9838	5	40	
	30	0.1822	29	0.1853	30	5.3955	862	0.9833	6	30	
	40	0.1851	29	0.1883	31	5.3093	836	0.9827	5	20	
	50	0.1880	28	0.1914	30	5.2257	811	0.9822	6	10	
11	0	0.1908	29	0.1944	30	5.1446	788	0.9816	5	0	79
	10	0.1937	28	0.1974	30	5.0658	764	0.9811	6	50	
	20	0.1965	29	0.2004	31	4.9894	742	0.9805	6	40	
	30	0.1994	28	0.2035	30	4.9152	722	0.9799	6	30	
	40	0.2022	29	0.2065	30	4.8430	701	0.9793	6	20	
	50	0.2051	28	0.2095	31	4.7729	683	0.9787	6	10	
12	0	0.2079	29	0.2126	30	4.7046	664	0.9781	6	0	78
	10	0.2108	28	0.2156	30	4.6382	646	0.9775	6	50	
	20	0.2136	28	0.2186	31	4.5736	629	0.9769	6	40	
	30	0.2164	29	0.2217	30	4.5107	613	0.9763	6	30	
	40	0.2193	28	0.2247	31	4.4494	597	0.9757	7	20	
	50	0.2221	29	0.2278	31	4.3897	582	0.9750	6	10	
13	0	0.2250	28	0.2309	30	4.3315	568	0.9744	7	0	77
	10	0.2278	28	0.2339	31	4.2747	554	0.9737	7	50	
	20	0.2306	28	0.2370	31	4.2193	540	0.9730	6	40	
	30	0.2334	29	0.2401	31	4.1653	527	0.9724	7	30	
	40	0.2363	28	0.2432	30	4.1126	515	0.9717	7	20	
	50	0.2391	28	0.2462	31	4.0611	503	0.9710	7	10	
14	0	0.2419	28	0.2493	31	4.0108	491	0.9703	7	0	76
	10	0.2447	29	0.2524	31	3.9617	481	0.9696	7	50	
	20	0.2476	28	0.2555	31	3.9136	469	0.9689	8	40	
	30	0.2504	28	0.2586	31	3.8667	459	0.9681	7	30	
	40	0.2532	28	0.2617	31	3.8208	448	0.9674	7	20	
	50	0.2560	28	0.2648	31	3.7760	439	0.9667	8	10	
15	0	0.2588	28	0.2679	32	3.7321	430	0.9659	7	0	75
	10	0.2616	28	0.2711	31	3.6891	421	0.9652	8	50	
	20	0.2644	28	0.2742	31	3.6470	411	0.9644	8	40	
	30	0.2672	28	0.2773	32	3.6059	403	0.9636	8	30	
	40	0.2700	28	0.2805	31	3.5656	395	0.9628	7	20	
	50	0.2728	28	0.2836	31	3.5261	387	0.9621	8	10	
16	0	0.2756	28	0.2867	32	3.4874	379	0.9613	8	0	74
	10	0.2784	28	0.2899	32	3.4495	371	0.9605	9	50	
	20	0.2812	28	0.2931	31	3.4124	365	0.9596	8	40	
	30	0.2840	28	0.2962	32	3.3759	357	0.9588	8	30	
	40	0.2868	28	0.2994	32	3.3402	350	0.9580	8	20	
	50	0.2896	28	0.3026	31	3.3052	343	0.9572	9	10	
17	0	0.2924	28	0.3057	32	3.2709	338	0.9563	8	0	73
	10	0.2952	27	0.3089	32	3.2371	330	0.9555	9	50	
	20	0.2979	28	0.3121	32	3.2041	325	0.9546	9	40	
	30	0.3007	28	0.3153	32	3.1716	319	0.9537	9	30	
	40	0.3035	27	0.3185	32	3.1397	313	0.9528	8	20	
	50	0.3062	28	0.3217	32	3.1084	307	0.9520	9	10	
18	0	0.3090		0.3249		3.0777		0.9511		0	72
		Cos.	d.	Cot.	d.	Tan.	d.	Sin.	d.	′	°

P. P.

	32	31	30
1	3.2	3.1	3.0
2	6.4	6.2	6.0
3	9.6	9.3	9.0
4	12 8	12.4	12.0
5	16.0	15.5	15.0
6	19.2	18.6	18.0
7	22.4	21.7	21.0
8	25.6	24.8	24.0
9	28.8	27.9	27.0

	29	28	27
1	2.9	2.8	2.7
2	5.8	5.6	5.4
3	8.7	8.4	8.1
4	11.6	11.2	10.8
5	14.5	14.0	13.5
6	17.4	16.8	16.2
7	20.3	19.6	18.9
8	23.2	22.4	21.6
9	26.1	25.2	24.3

	9	8
1	0.9	0.8
2	1.8	1.6
3	2.7	2.4
4	3.6	3.2
5	4.5	4.0
6	5.4	4.8
7	6.3	5.6
8	7.2	6.4
9	8.1	7.2

	7	6
1	0.7	0.6
2	1.4	1.2
3	2.1	1.8
4	2.8	2.4
5	3.5	3.0
6	4.2	3.6
7	4.9	4.2
8	5.6	4.8
9	6.3	5.4

	5	4
1	0.5	0.4
2	1.0	0.8
3	1.5	1.2
4	2.0	1.6
5	2.5	2.0
6	3.0	2.4
7	3.5	2.8
8	4.0	3.2
9	4.5	3.6

P. P.

°	′	Sin.	d.	Tan.	d.	Cot.	d.	Cos.	d.		
18	0	0.3090	28	0.3249	32	3.0777	302	0.9511	9	0	72
	10	0.3118	27	0.3281	33	3.0475	297	0.9502	10	50	
	20	0.3145	28	0.3314	32	3.0178	291	0.9492	9	40	
	30	0.3173	28	0.3346	32	2.9887	287	0.9483	9	30	
	40	0.3201	27	0.3378	33	2.9600	281	0.9474	9	20	
	50	0.3228	28	0.3411	32	2.9319	277	0.9465	10	10	
19	0	0.3256	27	0.3443	33	2.9042	272	0.9455	9	0	71
	10	0.3283	28	0.3476	32	2.8770	268	0.9446	10	50	
	20	0.3311	27	0.3508	33	2.8502	263	0.9436	10	40	
	30	0.3338	27	0.3541	33	2.8239	259	0.9426	9	30	
	40	0.3365	28	0.3574	33	2.7980	255	0.9417	10	20	
	50	0.3393	27	0.3607	33	2.7725	250	0.9407	10	10	
20	0	0.3420	28	0.3640	33	2.7475	247	0.9397	10	0	70
	10	0.3448	27	0.3673	33	2.7228	243	0.9387	10	50	
	20	0.3475	27	0.3706	33	2.6985	239	0.9377	10	40	
	30	0.3502	27	0.3739	33	2.6746	235	0.9367	11	30	
	40	0.3529	28	0.3772	33	2.6511	232	0.9356	10	20	
	50	0.3557	27	0.3805	34	2.6279	228	0.9346	10	10	
21	0	0.3584	27	0.3839	33	2.6051	225	0.9336	11	0	69
	10	0.3611	27	0.3872	34	2.5826	221	0.9325	10	50	
	20	0.3638	27	0.3906	33	2.5605	219	0.9315	11	40	
	30	0.3665	27	0.3939	34	2.5386	214	0.9304	11	30	
	40	0.3692	27	0.3973	33	2.5172	212	0.9293	10	20	
	50	0.3719	27	0.4006	34	2.4960	209	0.9283	11	10	
22	0	0.3746	27	0.4040	34	2.4751	206	0.9272	11	0	68
	10	0.3773	27	0.4074	34	2.4545	203	0.9261	11	50	
	20	0.3800	27	0.4108	34	2.4342	200	0.9250	11	40	
	30	0.3827	27	0.4142	34	2.4142	197	0.9239	11	30	
	40	0.3854	27	0.4176	34	2.3945	195	0.9228	12	20	
	50	0.3881	26	0.4210	35	2.3750	191	0.9216	11	10	
23	0	0.3907	27	0.4245	34	2.3559	190	0.9205	11	0	67
	10	0.3934	27	0.4279	35	2.3369	186	0.9194	12	50	
	20	0.3961	26	0.4314	34	2.3183	185	0.9182	11	40	
	30	0.3987	27	0.4348	35	2.2998	181	0.9171	12	30	
	40	0.4014	27	0.4383	34	2.2817	180	0.9159	12	20	
	50	0.4041	26	0.4417	35	2.2637	177	0.9147	12	10	
24	0	0.4067	27	0.4452	35	2.2460	174	0.9135	11	0	66
	10	0.4094	26	0.4487	35	2.2286	173	0.9124	12	50	
	20	0.4120	27	0.4522	35	2.2113	170	0.9112	12	40	
	30	0.4147	26	0.4557	35	2.1943	168	0.9100	12	30	
	40	0.4173	27	0.4592	36	2.1775	166	0.9088	13	20	
	50	0.4200	26	0.4628	35	2.1609	164	0.9075	12	10	
25	0	0.4226	27	0.4663	36	2.1445	162	0.9063	12	0	65
	10	0.4253	26	0.4699	35	2.1283	160	0.9051	13	50	
	20	0.4279	26	0.4734	36	2.1123	158	0.9038	12	40	
	30	0.4305	26	0.4770	36	2.0965	156	0.9026	13	30	
	40	0.4331	27	0.4806	35	2.0809	154	0.9013	12	20	
	50	0.4358	26	0.4841	36	2.0655	152	0.9001	13	10	
26	0	0.4384	26	0.4877	36	2.0503	150	0.8988	13	0	64
	10	0.4410	26	0.4913	37	2.0353	149	0.8975	13	50	
	20	0.4436	26	0.4950	36	2.0204	147	0.8962	13	40	
	30	0.4462	26	0.4986	36	2.0057	145	0.8949	13	30	
	40	0.4488	26	0.5022	37	1.9912	144	0.8936	13	20	
	50	0.4514	26	0.5059	36	1.9768	142	0.8923	13	10	
27	0	0.4540		0.5095		1.9626		0.8910		0	63
		Cos.	d.	Cot.	d.	Tan.	d.	Sin.	d.	′	°

P. P.

	37	36	35
1	3.7	3.6	3.5
2	7.4	7.2	7.0
3	11.1	10.8	10.5
4	14.8	14.4	14.0
5	18.5	18.0	17.5
6	22.2	21.6	21.0
7	25.9	25.2	24.5
8	29.6	28.8	28.0
9	33.3	32.4	31.5

	34	33	32
1	3.4	3.3	3.2
2	6.8	6.6	6.4
3	10.2	9.9	9.6
4	13.6	13.2	12.8
5	17.0	16.5	16.0
6	20.4	19.8	19.2
7	23.8	23.1	22.4
8	27.2	26.4	25.6
9	30.6	29.7	28.8

	28	27	26
1	2.8	2.7	2.6
2	5.6	5.4	5.2
3	8.4	8.1	7.8
4	11.2	10.8	10.4
5	14.0	13.5	13.0
6	16.8	16.2	15.6
7	19.6	18.9	18.2
8	22.4	21.6	20.8
9	25.2	24.3	23.4

	13	12
1	1.3	1.2
2	2.6	2.4
3	3.9	3.6
4	5.2	4.8
5	6.5	6.0
6	7.8	7.2
7	9.1	8.4
8	10.4	9.6
9	11.7	10.8

	11	10	9
1	1.1	1.0	0.9
2	2.2	2.0	1.8
3	3.3	3.0	2.7
4	4.4	4.0	3.6
5	5.5	5.0	4.5
6	6.6	6.0	5.4
7	7.7	7.0	6.3
8	8.8	8.0	7.2
9	9.9	9.0	8.1

P. P.

°	′	Sin.	d.	Tan.	d.	Cot.	d.	Cos.	d.	′	°
27	0	0.4540		0.5095		1.9626		0.8910		0	63
	10	0.4566	26	0.5132	37	1.9486	140	0.8897	13	50	
	20	0.4592	26	0.5169	37	1.9347	139	0.8884	13	40	
	30	0.4617	25	0.5206	37	1.9210	137	0.8870	14	30	
	40	0.4643	26	0.5243	37	1.9074	136	0.8857	13	20	
	50	0.4669	26	0.5280	37	1.8940	134	0.8843	14	10	
28	0	0.4695	26	0.5317	37	1.8807	133	0.8829	14	0	62
	10	0.4720	25	0.5354	37	1.8676	131	0.8816	13	50	
	20	0.4746	26	0.5392	38	1.8546	130	0.8802	14	40	
	30	0.4772	26	0.5430	38	1.8418	128	0.8788	14	30	
	40	0.4797	25	0.5467	37	1.8291	127	0.8774	14	20	
	50	0.4823	26	0.5505	38	1.8165	126	0.8760	14	10	
29	0	0.4848	25	0.5543	38	1.8040	125	0.8746	14	0	61
	10	0.4874	26	0.5581	38	1.7917	123	0.8732	14	50	
	20	0.4899	25	0.5619	38	1.7796	121	0.8718	14	40	
	30	0.4924	25	0.5658	39	1.7675	121	0.8704	14	30	
	40	0.4950	26	0.5696	38	1.7556	119	0.8689	15	20	
	50	0.4975	25	0.5735	39	1.7437	119	0.8675	14	10	
30	0	0.5000	25	0.5774	39	1.7321	116	0.8660	15	0	60
	10	0.5025	25	0.5812	38	1.7205	116	0.8646	14	50	
	20	0.5050	25	0.5851	39	1.7090	115	0.8631	15	40	
	30	0.5075	25	0.5890	39	1.6977	113	0.8616	15	30	
	40	0.5100	25	0.5930	40	1.6864	113	0.8601	15	20	
	50	0.5125	25	0.5969	39	1.6753	111	0.8587	14	10	
31	0	0.5150	25	0.6009	40	1.6643	110	0.8572	15	0	59
	10	0.5175	25	0.6048	39	1.6534	109	0.8557	15	50	
	20	0.5200	25	0.6088	40	1.6426	108	0.8542	15	40	
	30	0.5225	25	0.6128	40	1.6319	107	0.8526	16	30	
	40	0.5250	25	0.6168	40	1.6212	107	0.8511	15	20	
	50	0.5275	25	0.6208	40	1.6107	105	0.8496	15	10	
32	0	0.5299	24	0.6249	41	1.6003	104	0.8480	16	0	58
	10	0.5324	25	0.6289	40	1.5900	103	0.8465	15	50	
	20	0.5348	24	0.6330	41	1.5798	102	0.8450	15	40	
	30	0.5373	25	0.6371	41	1.5697	101	0.8434	16	30	
	40	0.5398	25	0.6412	41	1.5597	100	0.8418	16	20	
	50	0.5422	24	0.6453	41	1.5497	100	0.8403	15	10	
33	0	0.5446	24	0.6494	41	1.5399	98	0.8387	16	0	57
	10	0.5471	25	0.6536	42	1.5301	98	0.8371	16	50	
	20	0.5495	24	0.6577	41	1.5204	97	0.8355	16	40	
	30	0.5519	24	0.6619	42	1.5108	96	0.8339	16	30	
	40	0.5544	25	0.6661	42	1.5013	95	0.8323	16	20	
	50	0.5568	24	0.6703	42	1.4919	94	0.8307	16	10	
34	0	0.5592	24	0.6745	42	1.4826	93	0.8290	17	0	56
	10	0.5616	24	0.6787	42	1.4733	93	0.8274	16	50	
	20	0.5640	24	0.6830	43	1.4641	92	0.8258	16	40	
	30	0.5664	24	0.6873	43	1.4550	91	0.8241	17	30	
	40	0.5688	24	0.6916	43	1.4460	90	0.8225	16	20	
	50	0.5712	24	0.6959	43	1.4370	90	0.8208	17	10	
35	0	0.5736	24	0.7002	43	1.4281	89	0.8192	16	0	55
	10	0.5760	24	0.7046	44	1.4193	88	0.8175	17	50	
	20	0.5783	23	0.7089	43	1.4106	87	0.8158	17	40	
	30	0.5807	24	0.7133	44	1.4019	87	0.8141	17	30	
	40	0.5831	24	0.7177	44	1.3934	85	0.8124	17	20	
	50	0.5854	23	0.7221	44	1.3848	86	0.8107	17	10	
36	0	0.5878	24	0.7265	44	1.3764	84	0.8090	17	0	54
		Cos.	d.	Cot.	d.	Tan.	d.	Sin.	d.	′	°

P. P.

	44	43	42
1	4.4	4.3	4.2
2	8.8	8.6	8.4
3	13.2	12.9	12.6
4	17.6	17.2	16.8
5	22.0	21.5	21.0
6	26.4	25.8	25.2
7	30.8	30.1	29.4
8	35.2	34.4	33.6
9	39.6	38.7	37.8

	41	40	39
1	4.1	4.0	3.9
2	8.2	8.0	7.8
3	12.3	12.0	11.7
4	16.4	16.0	15.6
5	20.5	20.0	19.5
6	24.6	24.0	23.4
7	28.7	28.0	27.3
8	32.8	32.0	31.2
9	36.9	36.0	35.1

	38	37
1	3.8	3.7
2	7.6	7.4
3	11.4	11.1
4	15.2	14.8
5	19.0	18.5
6	22.8	22.2
7	26.6	25.9
8	30.4	29.6
9	34.2	33.3

	26	25	24
1	2.6	2.5	2.4
2	5.2	5.0	4.8
3	7.8	7.5	7.2
4	10.4	10.0	9.6
4	13.0	12.5	12.0
6	15.6	15.0	14.4
7	18.2	17.5	16.8
8	20.8	20.0	19.2
9	23.4	22.5	21.6

	23	17	16
1	2.3	1.7	1.6
2	4.6	3.4	3.2
3	6.9	5.1	4.8
4	9.2	6.8	6.4
5	11.5	8.5	8.0
6	13.8	10.2	9.6
7	16.1	11.9	11.2
8	18.4	13.6	12.8
9	20.7	15.3	14.4

	15	14	13
1	1.5	1.4	1.3
2	3.0	2.8	2.6
3	4.5	4.2	3.9
4	6.0	5.6	5.2
5	7.5	7.0	6.5
6	9.0	8.4	7.8
7	10.5	9.8	9.1
8	12.0	11.2	10.4
9	13.5	12.6	11.7

P. P.

°	′	Sin.	d.	Tan.	d.	Cot.	d.	Cos.	d.	′	°
36	0	0.5878	23	0.7265	45	1.3764	84	0.8090	17	0	54
	10	0.5901	24	0.7310	45	1.3680	83	0.8073	17	50	
	20	0.5925	23	0.7355	45	1.3597	83	0.8056	17	40	
	30	0.5948	24	0.7400	45	1.3514	82	0.8039	18	30	
	40	0.5972	23	0.7445	45	1.3432	81	0.8021	17	20	
	50	0.5995	23	0.7490	46	1.3351	81	0.8004	18	10	
37	0	0.6018	23	0.7536	45	1.3270	80	0.7986	17	0	53
	10	0.6041	24	0.7581	46	1.3190	79	0.7969	18	50	
	20	0.6065	23	0.7627	46	1.3111	79	0.7951	17	40	
	30	0.6088	23	0.7673	47	1.3032	78	0.7934	18	30	
	40	0.6111	23	0.7720	46	1.2954	78	0.7916	18	20	
	50	0.6134	23	0.7766	47	1.2876	77	0.7898	18	10	
38	0	0.6157	23	0.7813	47	1.2799	76	0.7880	18	0	52
	10	0.6180	22	0.7860	47	1.2723	76	0.7862	18	50	
	20	0.6202	23	0.7907	47	1.2647	75	0.7844	18	40	
	30	0.6225	23	0.7954	48	1.2572	75	0.7826	18	30	
	40	0.6248	23	0.8002	48	1.2497	74	0.7808	18	20	
	50	0.6271	22	0.8050	48	1.2423	74	0.7790	19	10	
39	0	0.6293	23	0.8098	48	1.2349	73	0.7771	18	0	51
	10	0.6316	22	0.8146	49	1.2276	73	0.7753	18	50	
	20	0.6338	23	0.8195	48	1.2203	72	0.7735	19	40	
	30	0.6361	22	0.8243	49	1.2131	72	0.7716	18	30	
	40	0.6383	23	0.8292	50	1.2059	71	0.7698	19	20	
	50	0.6406	22	0.8342	49	1.1988	70	0.7679	19	10	
40	0	0.6428	22	0.8391	50	1.1918	71	0.7660	18	0	50
	10	0.6450	22	0.8441	50	1.1847	69	0.7642	19	50	
	20	0.6472	22	0.8491	50	1.1778	70	0.7623	19	40	
	30	0.6494	23	0.8541	50	1.1708	68	0.7604	19	30	
	40	0.6517	22	0.8591	51	1.1640	69	0.7585	19	20	
	50	0.6539	22	0.8642	51	1.1571	67	0.7566	19	10	
41	0	0.6561	22	0.8693	51	1.1504	68	0.7547	19	0	49
	10	0.6583	21	0.8744	52	1.1436	67	0.7528	19	50	
	20	0.6604	22	0.8796	51	1.1369	66	0.7509	19	40	
	30	0.6626	22	0.8847	52	1.1303	66	0.7490	20	30	
	40	0.6648	22	0.8899	53	1.1237	66	0.7470	19	20	
	50	0.6670	21	0.8952	52	1.1171	65	0.7451	20	10	
42	0	0.6691	22	0.9004	53	1.1106	65	0.7431	19	0	48
	10	0.6713	21	0.9057	53	1.1041	64	0.7412	20	50	
	20	0.6734	22	0.9110	53	1.0977	64	0.7392	19	40	
	30	0.6756	21	0.9163	54	1.0913	63	0.7373	20	30	
	40	0.6777	22	0.9217	54	1.0850	64	0.7353	20	20	
	50	0.6799	21	0.9271	54	1.0786	62	0.7333	19	10	
43	0	0.6820	21	0.9325	55	1.0724	63	0.7314	20	0	47
	10	0.6841	21	0.9380	55	1.0661	62	0.7294	20	50	
	20	0.6862	22	0.9435	55	1.0599	61	0.7274	20	40	
	30	0.6884	21	0.9490	55	1.0538	61	0.7254	20	30	
	40	0.6905	21	0.9545	56	1.0477	61	0.7234	20	20	
	50	0.6926	21	0.9601	56	1.0416	61	0.7214	21	10	
44	0	0.6947	20	0.9657	56	1.0355	60	0.7193	20	0	46
	10	0.6967	21	0.9713	57	1.0295	60	0.7173	20	50	
	20	0.6988	21	0.9770	57	1.0235	59	0.7153	20	40	
	30	0.7009	21	0.9827	57	1.0176	59	0.7133	21	30	
	40	0.7030	20	0.9884	58	1.0117	59	0.7112	20	20	
	50	0.7050	21	0.9942	58	1.0058	58	0.7092	21	10	
45	0	0.7071		1.0000		1.0000		0.7071		0	45
		Cos.	d.	Cot.	d.	Tan.	d.	Sin.	d.	′	°

P. P.

	58	57	56	55
1	5.8	5.7	5.6	5.5
2	11.6	11.4	11.2	11.0
3	17.4	17.1	16.8	16.5
4	23.2	22.8	22.4	22.0
5	29.0	28.5	28.0	27.5
6	34.8	34.2	33.6	33.0
7	40.6	39.9	39.2	38.5
8	46.4	45.6	44.8	44.0
9	52.2	51.3	50.4	49.5

	54	53	52	51
1	5.4	5.3	5.2	5.1
2	10.8	10.6	10.4	10.2
3	16.2	15.9	15.6	15.3
4	21.6	21.2	20.8	20.4
5	27.0	26.5	26.0	25.5
6	32.4	31.8	31.2	30.6
7	37.8	37.1	36.4	35.7
8	43.2	42.4	41.6	40.8
9	48.6	47.7	46.8	45.9

	50	49	48
1	5.0	4.9	4.8
2	10.0	9.8	9.6
3	15.0	14.7	14.4
4	20.0	19.6	19.2
5	25.0	24.5	24.0
6	30.0	29.4	28.8
7	35.0	34.3	33.6
8	40.0	39.2	38.4
9	45.0	44.1	43.2

	47	46	45
1	4.7	4.6	4.5
2	9.4	9.2	9.0
3	14.1	13.8	13.5
4	18.8	18.4	18.0
5	23.5	23.0	22.5
6	28.2	27.6	27.0
7	32.9	32.2	31.5
8	37.6	36.8	36.0
9	42.3	41.4	40.5

	24	23	22	21
1	2.4	2.3	2.2	2.1
2	4.8	4.6	4.4	4.2
3	7.2	6.9	6.6	6.3
4	9.6	9.2	8.8	8.4
5	12.0	11.5	11.0	10.5
6	14.4	13.8	13.2	12.6
7	16.8	16.1	15.4	14.7
8	19.2	18.4	17.6	16.8
9	21.6	20.7	19.8	18.9

	20	19	18	17
1	2.0	1.9	1.8	1.7
2	4.0	3.8	3.6	3.4
3	6.0	5.7	5.4	5.1
4	8.0	7.6	7.2	6.8
5	10.0	9.5	9.0	8.5
6	12.0	11.4	10.8	10.2
7	14.0	13.3	12.6	11.9
8	16.0	15.2	14.4	13.6
9	18.0	17.1	16.2	15.3

P. P.

SURVEYING

INSTRUMENTS USED

THE COMPASS

Surveying is an extension of mensuration, and, as ordinarily practiced, may be divided into surface work, or *ordinary surveying*, and underground work, or *mine surveying*. With slight modifications, the instruments employed in both are the same.

The *compass* used may be either a pocket compass, or a surveyor's compass; it may be held in the hand, or on a tripod. The Jacob's staff is convenient for use on the surface, but is frequently useless in the mine. The compass is not accurate enough for the construction of a general map of the mine. It may be used to secure an approximate idea of the shape of the workings, so as to plan an approximate course on which to drive an opening designed to connect two or more given points. If the opening is one that will be expensive to drive, and should be straight, the compass survey should never be relied on.

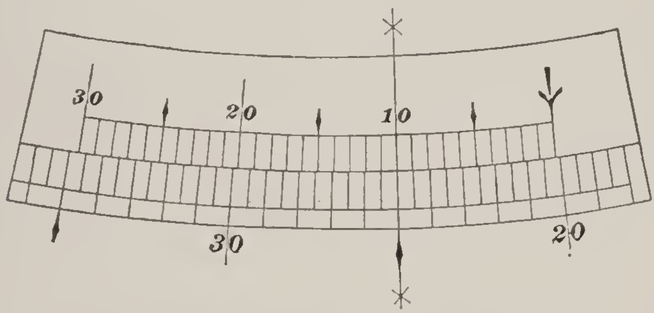
Using the Compass.—In using the compass, the surveyor should keep the south end toward his person, and read the bearings from the north end of the needle, care being taken always to keep the compass level. In the surveyor's compass, the position of the letters E and W are reversed from their natural position, in order that the direction of the sight may be correctly read. As the circle is graduated to $\frac{1}{2}°$, a little practice will enable the surveyor to read the bearings to quarters, estimating with his eye the space bisected by the point of the needle.

The compass is usually divided into quadrants, and 0 is placed at the north and south ends; 90° is placed at the E and W marks, and the graduations run right and left from the 0 to 90°. In reading the bearing, if the sights are pointed in a N W direction, the north end of the needle, which always points approximately north, is to the right of the front sight or front

end of the telescope, and, as the number of degrees is read from it, the letters marking the cardinal points of the compass read correctly. If the E, or east, mark were on the right side of the circle, a N W course would read N E. This same fact applies to all four quadrants.

THE TRANSIT

The *transit* is the only instrument that should be used for measuring angles in any survey where great accuracy is desired. Its advantages over a vernier compass are mainly due to the use of a telescope. With it angles can be measured either vertically or horizontally, and, as the vernier is used throughout, extreme accuracy is secured. In mine work all screws and movable parts should be covered, so as to keep out acid, water, and dust; if this is not done, the instrument will soon be destroyed. The vertical circle on the transit may be a full circle or a segment; the former is preferred, as it is always ready without intermediate clamp screws.



Transit Verniers.—The verniers on a transit differ from those on a compass in detail only; the principle is the same. The transit vernier is so divided that 30 spaces on it equal in length 29 on the limb of the instrument. It is read practically the same as a compass vernier, except that on the transit the vernier is made with all of the 30 divisions on one side of the 0 mark.

Each division of the vernier is, therefore, $\frac{1}{30}$, or, in other words, 1′ shorter than the $\frac{1}{2}$° graduations on the limb. In the figure the reading is 20° 10′. If the 0 on the vernier should be beyond $20\frac{1}{2}$° on the limb of the transit, and the line 10 should

coincide with a line on the limb, the reading would be 20° 40′. In case the 12th line from 0 should coincide with a line on the limb, the reading would be 20° 42′, etc.

In some transits, the graduated limb has two sets of concentric ,graduations, the 0 in both being the same. While the outside set is marked from 0° each way to 90°, and thence to 0° on the opposite side of the circle, the other set is marked from 0° to 360° to the right, as a clock face. The inside set has the N, S, E, and W points marked, the 0° of the inside set being taken as north.

Transit Telescopes.—The interior of the telescope is fitted with a diaphragm, or cross-wire ring, to which cross-wires are attached. These cross-wires are either platinum or strands of spider web. For inside work, platinum should be used, as spider web is translucent and cannot readily be seen. These wires are set at right angles to each other and are so arranged that one can be adjusted so as to be vertical and the other horizontal. This diaphragm is suspended in the telescope by four capstan-headed screws, and can be moved in either direction by working the screws with an ordinary adjusting pin. The intersection of the wires forms a very minute point, that, when adjusted, determines the optical axis of the telescope, and enables the surveyor to fix it upon an object with the greatest precision.

The imaginary line passing through the optical axis of the telescope is termed the *line of collimation*, and the operation of bringing the intersection of the wires into the optical axis is called the *adjustment of the line of collimation.*

The transit should not be subjected to sudden changes in temperature that may break the cross-hairs. In case of a break, the cross-hair diaphragm must be removed and the broken wire replaced.

CHAIN, TAPE, PINS, AND PLUMB-BOB

The *chain*, which is generally 50 or 100 ft. long, should be made of annealed steel wire, each link being exactly 1 ft. in length.

The *steel tape* is simply a ribbon of steel, on which are marked, by etching, or other means, the different graduations; these

may be inches or tenths of a foot, or every foot. It is wound on a reel, and may be any desired length up to 500 ft. When distances do not come at even feet, the fractional part of the foot should always be noted in tenths. Thus, 53 ft. 6 in. should always be noted as 53.5 ft.

For the most exact work steel tapes are now almost exclusively used by the leading mining engineers, on account of their greater accuracy as compared with chains.

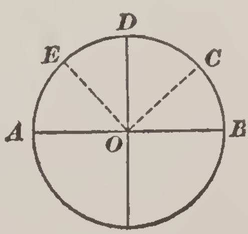
Pins should be from 15 to 18 in. long, made of tempered-steel wire, and should be pointed at one end and turned with a ring for a handle.

The *plumb-bob* takes the place of the transit rod in underground work, as the stations are usually in the roof, and strings are hung from them to furnish foresights and backsights. Plumb-bobs vary in weight and shape. The cord is best illuminated by placing white paper or cardboard behind it and holding the lamp in front and to one side. The string shows as a dark line against a white ground, and there is less difficulty in finding it than when the light is placed exactly behind it.

The *clinometer*, or *slope level*, is a valuable instrument for side-note work, but it is not accurate enough for a survey; its place has been taken by the vertical circle on the transit. Clinometers are of two styles, one showing the inclination by means of a bubble and the other, by means of a pendulum. The latter is the old-fashioned and more accurate German Gradbogen.

TRANSIT SURVEYING

READING ANGLES



The angle read may be included or deflected. If the transit is set up at O with backsight at B and foresight at C, there are two angles made by the line $C\ O$ with the line $B\ O\ A$, namely the *included angle* $B\ O\ C$, and the *deflected angle* $C\ O\ A$.

To read the included angle set the zeros of the vernier and graduated limb together accurately, and clamp the plates.

Turn the telescope on the backsight, with the level bubble down, and, when set, fasten the lower clamp so as to fix both clamped plates to the tripod head. Loosen the upper clamp, turn the telescope to *C*, and set accurately. The vernier will read, for example, 45° left angle.

To read the deflected angle arrange the verniers as before, being careful to turn the telescope over on its axis until the bubble tube is up, and then take the backsight and fix lower clamp. Turn the telescope back (this is called *plunging* the telescope) and then sight to foresight and fix as before; the vernier will then read a right angle of 135°. The sum of included and deflected angles must always be 180°.

NOTE.—In making a survey by included angles it is necessary to add or subtract 180° at each reading to have the vernier and compass agree; by deflected angles they will agree without the above addition or subtraction, therefore the latter method is generally used.

If the dip of a sight is to be taken the tape must be held at the transit head and stretched in the line of sight. If the pitch of the ground is to be taken the point of foresight must be at the same height as the axis of the transit and the sight will then be parallel to the surface. The angle of dip is read "plus" or "minus" as it is above or below the horizontal plane. If we have the dip of a sight and the distance between the transit head and the point of sight we can get the vertical and horizontal components of that distance from the table of sines and cosines.

MAKING SURVEY WITH TRANSIT

Surveying by Means of Individual Angles.—To survey by means of individual angles, set the vernier at 0 of limb, plunge the telescope, and, when set on the backsight, loosen the needle and read the bearing of the line from backsight to set-up. Plunge the telescope back and set on the foresight and read both the needle and the vernier. The difference in the needle readings should agree with the vernier reading within 15′, as local attraction will affect the needle equally on both sights.

NOTE.—As the moving of any mass of iron or steel during the readings of the needle will affect the same and destroy the value of the needle as a check, the tape and other iron materials should not be moved during the taking of angles.

Surveying by Means of Continuous Vernier.—To survey by means of continuous angles, set the vernier at 0, unclamp the

compass needle, and, when stationary, turn the north point of the compass limb so as to coincide with the north point of the needle. Fix the lower clamp, plunge the telescope, and take a backsight by loosening the upper clamp. The vernier and the needle should agree in giving the magnetic bearing of the line from backsight to set-up. Record this in the notebook; plunge the telescope, and take a foresight; the needle and the vernier should agree as before. After making the record, set up over the foresight and take a sight to the station just left with the telescope plunged, having first seen that the vernier reads exactly as it did on the last foresight, as a slip in carrying the transit from one station to another, which is not detected at the time, can never be checked afterwards when the final work is found to be in error. The foresight is taken as before; on every sight the needle and the vernier should agree if there is no local attraction of the needle.

If all the corners of a field that is to be surveyed can be seen, from a central point, the survey can be made by setting up the transit at that point, and, with one corner as a backsight, taking all the other corners as foresights with but one set-up, and measuring from this point to all of the corners; or the transit can be set up at any corner and a line run around the field. This latter method is called *meandering*. Both methods will give the same result when plotted.

PLOTTING

A *plot* is a map drawn to a given scale, and showing all of the natural features. *Plotting* is the making of such a map from notes of a survey, and may or may not require the permanent placing on it of the stations, by which the survey is made. In underground work, the exact location and the retention of those stations is of the first importance, and is secondary only to the exact plotting of the side notes. The scale of the plot is generally as large as will show the points of interest in the property; but in Pennsylvania, the maps for coal mines must be drawn to a scale of 100 ft. = 1 in. There are two methods of plotting: by protractor, and by coordinates. When the scale is sufficiently large, it makes little difference which method is used, if the work be carefully done

with exact instruments; but with small scales, 100 ft. = 1 in., or smaller, the method by coordinates should be used. When the scale is from 1 to 25 ft. to 1 in., the errors are small enough to make little chances of variation in a close of ten or twelve stations; when the survey is of short sights from a main line to points where no further work is to be done, the protractor will afford a quick method of plotting.

To Calculate Vertical Distances.—When making the survey, read the vertical angles to all stations. If the angle is one of depression, place a minus sign (−) before it; if it is an angle of elevation, place a plus sign (+) before it. These will show whether the vertical distance is to be added to, or subtracted from the height of the preceding station.

Having the horizontal distance and the vertical angle:

Distance × tangent of vertical angle = vertical distance.

Having the pitch distance and vertical angle:

Distance × sine of vertical angle = vertical distance.

To Calculate Horizontal Distance, or Latitude.—Pitch distance × cosine of vertical angle = horizontal distance.

Vertical height or departure ÷ tangent of vertical angle = horizontal distance.

To Calculate Pitch Distance.—Horizontal distance ÷ cosine of bearing, or multiplied by secant of bearing = pitch distance.

Vertical distance ÷ sine of vertical angle, or multiplied by cosecant of bearing = pitch distance.

To Calculate Vertical Angle.—Horizontal distance ÷ the pitch distance = cosine of vertical angle.

Vertical distance ÷ pitch distance = sine of vertical angle.

Vertical distance ÷ horizontal distance = tangent of vertical angle.

NOTE.—Whenever sines, cosines, tangents, etc., are here named, they mean the natural sines, etc. of the angle.

Plotting by Coordinates.—In the establishment of a meridian and a fixed point, the latter should be a stone post, or iron plug sunk in solid rock; this point is called the *origin of coordinates.* Have the principal meridian passing through this point in an exact north-and-south direction, and a secondary meridian or base line passing through this point at right angles to the first, or in an exact east-and-west line. Any point on the map will

then be a certain distance north or south, and east or west of the origin. The lines drawn from this point at right angles to the two base lines just given are called the *coordinates of that point*, and the point can be plotted when they are given. For example, the coordinates of a station are N 345.67, and E 890.12. Measure 890.12 ft. east of the origin on the secondary meridian and, from this point, 345.67 ft. north to the point desired. Or measure on the primary meridian to the north and then turn off a right angle to the east and reach the same point. In any event the position of each station may be plotted independently of all the others, and any error in locating one is not carried to the next. When two stations are plotted, the distance between them on the map should be exactly what is found for their horizontal distance on the ground. This check shows whether the plotting is correct. This is also called *traversing* a survey if the meridian is north and south, and in books on surveying there are printed *traverse tables*, which are accurate within certain limits, but not so accurate as the *tables of coordinates* published separately, as the latter are carried to a greater number of decimals.

With a north-and-south meridian, the point from which the measuring of the angles is begun, the zero point, is the north point, and the angles are read for continuous vernier in the direction of the hands of a watch. The sines of angles are *eastings* and *westings*, and the cosines are *northings* and *southings*.

TRAVERSING A SURVEY

To traverse a survey, means to determine by calculation how far north or south and east or west any station may be from another, the location of which is fixed. To do this, all distances must be measured horizontally, or calculated to horizontal distances. The horizontal angles, or courses, must be read as quadrant courses, or reduced from azimuth to quadrant courses. An *azimuth course* is one that is read on a transit that is graduated from 0° to 360°. A *quadrant course* is one read in the quadrant of the circle, as S 67° W, N 43° E, etc.

Latitude means distance north or south, and is determined by the first initial of the recorded course; thus, if a course is

S 67° W, the latitude is south; if N 43° E, the latitude is north. The latitude = distance × cosine of bearing.

Departure means distance east or west, and is determined by the last initial of the recorded course; thus, if a course is S 67° W, the departure is west; if N 43° E, the departure is east. The departure = distance × sine of bearing.

If the survey is a continuous one around a tract, and ending at the place of beginning, the sum of the northings should equal the sum of the southings, and the sum of the eastings should equal the sum of the westings. The most accurate way to construct a map is to traverse the survey and place all stations on it by the traversed distances, or to at least put a number of the principal stations on the map by the traversed distances, and use the protractor to plot only the intermediate stations.

DETERMINING AREA OF TRACT OF LAND

If the tract of land is a regular polygon, find the area by the rule given under the head of Mensuration for polygons of the same number of sides. If it is an irregular polygon, divide it into triangles and calculate the area of each triangle; the sum of these areas will be the area of the tract. If the tract is an irregular polygon in shape, the map should be made on as large a scale as possible, and the distances should be measured with the greatest care, owing to liability to error through very slight inaccuracies of measurement.

DETERMINING CONTENTS OF COAL SEAM

If the seam lies flat, multiply the area of the tract, in square feet, by the thickness of the seam, in feet; the product will be the cubic contents of the seam, in feet. If the seam is an inclined one, find its area by measuring the width of the tract on its line of pitch, and find the distance on the pitch of the seam by dividing the horizontal distance measured by the cosine of the angle of inclination; this will give the *pitch distance.* Multiply the pitch distance by the length of the tract, to find the area of the seam; this multiplied by its thickness will give the contents.

$$\text{Tons of coal} = \frac{\text{cubic contents, in feet} \times \text{sp. gr.} \times 62.5}{2{,}240}$$

UNDERGROUND SURVEYING

ESTABLISHMENT OF STATIONS

There are a number of variations in the foregoing practice that are caused by the entirely different set of conditions in underground work. As the establishment of stations is the most important duty of an engineer in surface work, so it takes the first place in work underground, as the accuracy of the work depends on the location of the stations, while its rapidity depends on using the least number consistent with completeness. Also, the fewer the number of stations, the less are the chances of error. In underground work, stations should be located under the conditions of permanence, freedom from destroying agencies, and ease of access. Temporary stations for a single sight need not fill all these requirements. They are generally established in the roof of the mine, less frequently in the floor. In the former case, a *center* must be established before each set-up of the transit. Places are chosen that will be least affected by subsequent work, and the stations are put in collars, lids, or wedges of props, in the props themselves when they have sufficient incline to allow the transit to be set under them, or in the roof itself. Wherever set, they should not project far from the surface, and thus be liable to be brushed away in a low gangway by cars with topping higher than usual, or knocked away by flying fragments from a shot, if near the working faces. Top stations have a mark about them to call attention to their location; it is generally a circle. When there are other corps at work in the same mine, the stations of the two surveys should be given distinguishing marks to avoid confusion.

Kinds of Stations.—The simplest top station is called by some a *jigger station.* It is a shallow conical hole, made with the point of the foresight man's hatchet which is dug into the top rock and rotated. The sights are given and the centers set by putting the plummet cord in this groove, and placing the end in the jigger hole in the roof. Common shingle nails are sometimes driven into collars, or cracks in the roof, and the end of the plummet line is noosed and put over the head.

This causes an eccentric hanging of the plummet that may result in an error in backsight and foresight of the width of the nail head, which will be quite appreciable in a short sight.

A wooden plug is driven into a hole drilled in the roof, and into this is driven the spad. The swelling wood clamps the same and prevents it from coming out as readily as it was put in.

A hole is bored in the roof with a $\frac{3}{32}$-in. twist drill and a piece of cord or a copper wire, placed across this and driven into the hole by a hardwood shoe peg. The plummet is tied to the lower end. A cord will soon rot, and, if in the gangway, will be pulled out by the drivers for whip lashes; while the wire is more permanent, it may be pulled out by catching in the topping of a car in a low place.

In the best form of station, however, the use of spads is dispensed with and all the stations are put in rock roof where possible, and consist of a vertical hole 1 in. deep made with a $\frac{3}{32}$-in. twist drill. When a sight is to be taken, the foresight man puts into this a steel clip with serrated edges; this clip is made by bending upon itself a thin piece of steel $\frac{3}{32}$-in. wide. When the ends are pressed together it will go into the hole, and the spring of the sides and the serrated edges hold it in place so that it is hard to pull out. The cord passes through a hole in the center of the bend and is, therefore, in the center of the hole, no matter how the clip is inserted. The clip is removed by pressing together the ends. This is the easiest and quickest way of working, as there is no eyehole to be freed from dirt and no knot to be tied and untied. The hanging of the plummet takes a fraction of a second, and the station will remain as long as the roof keeps up. The disadvantages are that holes may be bored inclined to the vertical by a careless man, and many roofs are unfit for piercing with a twist drill.

Marking Stations.—There should be some regular way of witnessing all stations. In general, a vertical line on the rib calls attention to a station in the floor near the side marked. A roof station has a mark around it, as has been described, and is some geometric figure. Each station must be lettered or numbered so that it can be readily recognized when the subsequent surveys are made.

White lead, or Dutch white, thinned with linseed oil, is ordinarily used for marking stations. The top should be wiped clean and dry with a piece of cotton waste before the paint is applied, or the white will be so discolored as to be scarcely visible, or the paint will flake off and the numbers will be lost.

Centers.—When the station is in the roof, there must be something for the transit to set over, as it is easier to do so than to set under a station, and much more accurate as instruments are now made. The set-up is made over a center. To avoid being displaced, centers are made as small and heavy as possible; they are usually made of lead in the following manner: A hole 1¼ in. in diameter and ½ in. deep is bored in a thick plank, and a brad is set in its center with the head down. The hole is filled with melted lead and the brad is slightly raised to surround the head with lead, and held with pincers in a vertical position until the lead has set. The brad is cut off ¼ in. above the lead and pointed. This center combines weight and small size,

KEEPING NOTES

Taking Notes.—Complete notes should be taken and recorded neatly and systematically, so that a stranger can easily follow them. Every physical characteristic and all surface improvements should be noted and located. Every ledge of rock should be noted, its character, dip, and course of strike should be taken. In a large company there should be a separate book for transit notes and for side notes, and where many collieries are operated, a separate set of books should be used for each colliery. But however the notes are kept, the following facts should be recorded: The numbers of the stations; the needle readings to check the vernier; the vernier reading; the dip of the sight; the distance measured, either flat or on the dip; the height of the axis of the transit from the ground; the height of the point sighted at from the ground; and all other necessary remarks to make the work plain. It is customary to have series of vertical columns headed (to suit the above) Sta., Needle F. S., Needle B. S., Vernier, Pitch, Dist., H. I. (height of instrument), H. R. (height of rod, or point to which sight was taken), and Remarks.

At the top of the page, in starting a survey, there should be entered the name of the mine and of the bed where the work is to be done; the names of the regular corps employed for the work, and those that were taken from the mine to point out work or assist; the instruments used; the date of the work, and, in case it is the continuation of a previous survey, the pages where such work was noted must be set down. Such books are complete records, and can be used as time books in paying the men, or as proofs of the kind of work done in case a lawsuit requires such testimony, by showing the number of men, the instruments used, and the time employed.

Transit and Side Notes.—There are about as many methods of keeping the transit and side notes as there are engineers. These methods arrange themselves into groups; those in most common use in the mines are:

The side notes of each sight follow the transit notes of that sight, and on the same page.

The side notes are entered in the same book on opposite pages.

The transit notes of the whole survey come first, and are followed by the side notes in the same book.

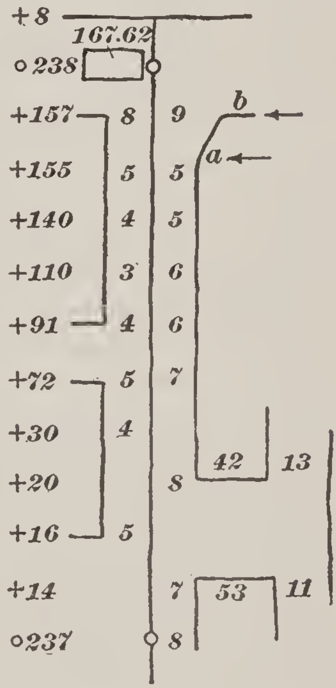
Each set of notes has a separate book.

Suppose that the transit is set up at *b*, with backsight at *a*; foresight to *c*, deflected angle $a\,b\,c = 85°\ 27'$ left, and that the distance *b c* is 421.76 ft. measured on a pitch of $+4°\ 35'$. Then some such form of notes as the following can be used. Other forms are used, but all are made to suit the ideas of the person surveying.

Sta.	Needle B. S.	Vernier		Needle F. S.	Pitch	Dist.	Sta.
		A	*B*				
− *a* *b*	S 25° 30′ W	L 85° 27′	L 85° 26′	S 60° 0′ E	+4° 35′	421.76	*c*

In every case the notes should convey to the man that plots some idea of the form of the place surveyed. An accurate sketch cannot be made unless the whole locality can be seen at a glance; it is not necessary to go to the other extreme and write down the notes without a sketch.

In the accompanying illustration, the red line in the center of the page of the notebook is taken as the line of survey, and the next parallel line on either side is taken as the boundaries of the solid on either side. The only figures on each side of the red line are the distances from the line to the solid, while the pluses at which they were taken are noted at the side of the page, and the exact distance between the two stations is enclosed in the parallelogram. This method at the pluses 155 and 157 calls attention to a point where practice varies greatly; namely, the noting of the corner pillar and the locating of the corner. One method calls the corner that point where the pillar begins to diverge from the gangway line, as at *a*, where a chamber, cross-cut, or counter starts from the gangway; a second method designates the corner as the first or last solid part met with in the line of survey, as at *b*. The first is faulty, as there is no record of the gradual divergence of the pillar from the gangway line, and the words "corner of pillar" usually mean the end of the same. The pillar should be called solid until the line at right angles to the line of survey is tangent to the ends, no matter whether that end is 10 or 100 ft. distant. Any one can plot side notes if accurately taken, and two persons accurately plotting such notes will reach the same result.



MINE CORPS

The method of dividing the work in an underground survey depends on the size of the corps, therefore the work of each man is considered in order to get the right number for the corps. The chief of the party should be where he can do the most good, and where he can plan the work for his subordinates. The principal point of the survey is the setting of the stations so as to do the work thoroughly with the fewest set-ups, and thus diminish the chances of error in instrument work. The chief should locate the stations and add all the necessary signs to show how the work is to be done. As the *transitman* should not have his attention distracted from his particular work by questions as to procedure, the chief should not run the transit. Upon this basis, the ideal mine corps consists of at least four, or better five, men from the office, and three from the mine. It is divided into two sections. The chief takes the men supplied by the mine—one or more of whom is acquainted with the work done since the last survey—and locates the stations; the transitman follows with the second section, to measure angles and distances. By this time the stations are set and the chief takes his men after the transit party and gets the side notes, with a check-measurement of the distances between stations.

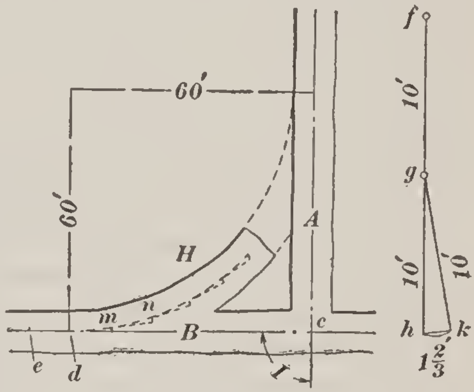
The *foresight man* should be intelligent and active, as the amount of work done in a day depends on his ability to keep ahead of the transitman. Some of the latter are fast enough to keep two foresight men on the jump. His duty is to set the center for the next set-up under the station, and also place the tripod if three are used in the work, to give the sight, and, in some corps, to carry the front end of the tape and assist in taking the distance.

The *backsight man* has little to do inside; for this reason, he cleans and oils the tape, gets out new plummet strings, and sees that the tools are ready for the next work, as soon as the corps gets to the office.

LAYING OUT CURVES IN A MINE

Curves in a mine are usually so sharp that they are designated as curves of so many feet radius, instead of as curves

of so many degrees. For example, suppose that it is required to connect the two headings *A* and *B*, in the accompanying illustration, which are perpendicular to each other, with a curve of 60 ft. radius. Prepare the device shown on the right-hand side, by taking three small wires or inelastic strings *f g*, *g h*, and *g k*, each 10 ft. long, and connecting one end of each to a small ring, and the other end of two to the ends of a piece of wood $1\frac{2}{3}$ ft. long. Form a neat loop at the end *f* of the third *g f*. To use this device, lay off on the center line of the heading *B*, *c d* and *d e* equal to 60 ft. and 10 ft., respectively. Place the loop *f* of the device over a small wire peg driven into the floor at *e*, and the ring *g* over a similar peg at *d*. Take hold of the stick *h k*, pull the strings *g h* and *g k* taut, and place the center mark on the piece of wood *hk* on the center line of the heading *B*. Drive a small peg in at *m*, located by the point *k*, which is on the curve. Move the device forwards, place the loop *f* over the peg at *d*, the ring *g* over the peg at *m*, and take hold of the stick *h k* and pull until the strings *g h* and *g k* are taut, and the strings *f g* and *g h* are in a straight line. The point *k* will fall on the curve at *n*, which mark by driving in a peg. To locate other points, proceed exactly as in the last step. The distance *c d* in any case is found by the formula



$$c\,d = R \tan \tfrac{1}{2} I$$

in which R = radius of curve;

I = intersection angle of center lines of headings

ELEMENTS OF MECHANICS

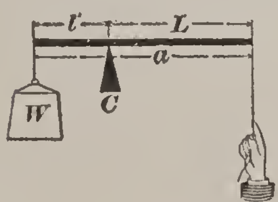
DEFINITIONS AND LAWS

All machinery, however complicated, is merely a combination of the six elementary forms: the *lever*, the *wheel and axle*, the *pulley*, the *inclined plane*, the *wedge*, and the *screw;* and these can be still further reduced to the lever and the inclined plane. They are termed *mechanical powers*, but they do not produce force; they are only methods of applying and directing it. The law of all mechanics is:

Law.—*The power multiplied by the distance through which it moves is equal to the weight multiplied by the distance through which it moves.*

Thus, 20 lb. of power moving through 5 ft. = 100 lb. of weight moving through 1 ft. In the following discussion friction is not considered, the idea being to give an elementary knowledge of the principles of the elements of mechanics.

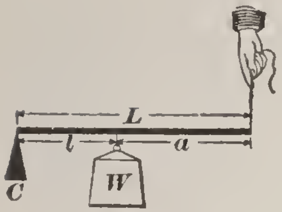
Levers.—There are three classes of levers. They are: (1) power at one end, weight at the other, and fulcrum between; (2) power at one end, fulcrum at the other, and weight between; (3) weight at one end, fulcrum at the other, and power between. A lever is in equilibrium when the arms balance each other. The distances through which the power and the weight move depend on the comparative length of the arms. Let L represent power's distance from the fulcrum (C), l the weight's distance, and a the distance between power and weight; then, if L is twice l, the power will move twice as far as the weight, Substituting these terms in the law of mechanics,



$$P:W = l:L \qquad PL = Wl$$

$$P = \frac{Wl}{L} \qquad W = \frac{PL}{l}$$

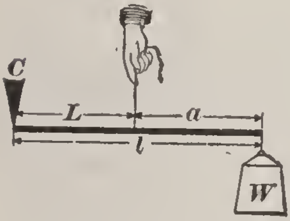
$$l = \frac{Pa}{W+P} \qquad L = \frac{Wa}{W+P}$$



$$P : W = l : L \qquad PL = Wl$$

$$P = \frac{Wl}{L} \qquad W = \frac{PL}{l}$$

$$L = \frac{Wa}{W-P} \qquad l = \frac{Pa}{W-P}$$



$$P : W = l : L \qquad PL = Wl$$

$$P = \frac{Wl}{L} \qquad W = \frac{PL}{l}$$

$$L = \frac{Wa}{P-W} \qquad l = \frac{Pa}{P-W}$$

In first- and second-class levers, as ordinarily used, power is gained but time is lost; and in the third class, time is gained but power is lost.

Example.—Having to lift a weight of 2,000 lb. with a lever, the short end of which is 2 ft. from the fulcrum and the long end 10 ft., how much power will be required?

Solution.—Substituting in the formula $L : l = P : W$ and solving gives $\frac{2{,}000 \times 2}{10} = 400$ lb.

Wheel and Axle.—The wheel and axle, Fig. 1, of which the ordinary windlass is a common form, is a modification of the lever. The power is applied to the handle, the bucket is the weight, and the axis of the windlass is the fulcrum. The long arm of the lever is the handle, and the short arm is the radius of the axle. Thus, F is the fulcrum, $F\,c$ the long arm, and $F\,b$ the short arm. The wheel and axle has the advantage that it is a kind of perpetual lever; it is not necessary to prop up the weight and readjust the lever, but both arms work continuously.

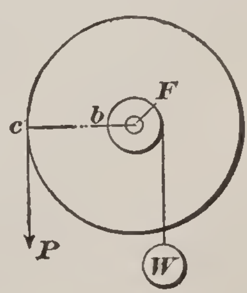


Fig. 1

By turning the handle or wheel around once, the rope will be wound once around the axle, and the weight will be lifted that distance. Applying the law of mechanics, power $\times$ circumference of wheel = weight $\times$ circumference of axle; or,

as the circumferences of circles are proportional to their radii,

$$P:W = r:R \qquad PR = Wr$$

$$P = \frac{Wr}{R} \qquad R = \frac{Wr}{P}$$

$$W = \frac{RP}{r} \qquad r = \frac{RP}{W}$$

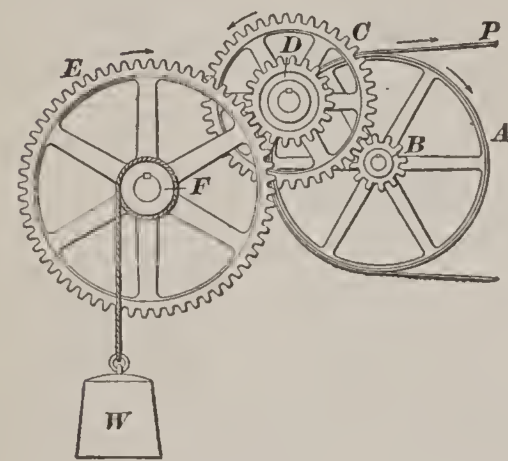


FIG. 2

A *train*, Fig. 2, consists of a series of wheels and axles that act on one another on the principle of a compound lever. The *driver* is the wheel to which power is applied. The *driven*, or *follower*, is the one that receives motion from the *driver*. The *pinion* is the small gear-wheel on the axle.

If the diameter of the wheel A is 16 in., and of the pinion B 4 in., a pull of 1 lb. applied at P will exert a force of 4 lb. on the wheel C. If the diameter of C is 6 in., and of D 3 in., a force of 4 lb. on C will exert a force of 8 lb. on E. If E is 16 in. in diameter, and F 4 in., a force of 8 lb. on E will raise a weight of 32 lb. on F. In order, however, to lift this amount according to the principle already named, the weight will only pass through $\frac{1}{32}$ of the distance of the power. Thus, power is gained and speed lost. To reverse this, apply power to the axle F, and, with a correspondingly heavy power, gain speed. Referring to Fig. 3, and applying the law of mechanics, the following formulas are obtained,

$$P = \frac{Wrr'r''}{RR'R''} \qquad W = \frac{PRR'R''}{rr'r''}$$

$$n:n'' = r'r'' : RR'$$

$$v:v' = rr'r'' : RR'R''$$

in which n, n', n'' = number of revolutions;
v, v' = velocity of speed of rotation;
r, r', r'', etc. = radii of the pinions;
R, R', R'', etc. = radii of the wheels.

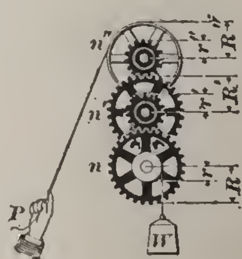


FIG. 3

Inclined Planes.—In the inclined plane, Fig. 4, the power must descend a distance equal to $A\,C$ to elevate the weight to the height $B\,C$; hence $P \times$length of inclined plane$=W$ $\times$height of inclined plane, or $P:W$ =height of inclined plane : length of inclined plane; or,

$$P=\frac{Wh}{l} \quad W=\frac{Pl}{h}=\frac{P}{\sin a}$$

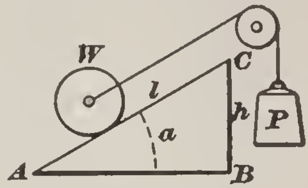


FIG. 4

Wedge.—The wedge usually consists of two inclined planes placed back to back. In theory, the same formula applies to the wedge as to the inclined plane.

$P:W=$thickness of wedge : length of wedge

Screws.—The screw consists of an inclined plane wound around a cylinder. The inclined plane forms the thread, and the cylinder, the body. It works in a nut that is fitted with reverse threads to move on the thread of the screw. The nut may run on the screw, or the screw in the nut. The power may be applied to either, as desired, by means of a wrench or a lever.

Pulleys.—The pulley is simply another form of the lever that turns about a fixed axis or fulcrum. With a single fixed pulley, shown in Fig. 5, there can be no gain of power or speed, as the force P must pull down as much as the weight W, and both move with the same velocity. It is simply a lever of the first class with equal arms, and is used to change the direction of the force. $v=$velocity of W; $v'=$velocity of P; $P=W$; $v=v'$.

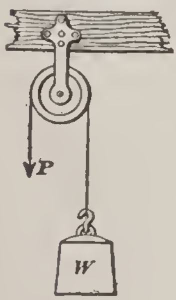


FIG. 5

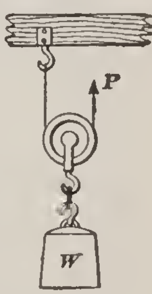


FIG. 6

In the *movable pulley*, shown in Fig. 6, one-half of the weight is sustained by the hook, and the other half by the power. As the power is only one-half the weight, it must move through twice the space; in other words, by taking twice the time, it is possible to lift twice as much. Here power is gained and time lost. $P=\frac{1}{2}W$; $v'=2v$.

In the *combination pulley*, shown in Fig. 7, the weight W is sustained by three cords, each of which is stretched by a tension equal to the power P, hence, 1 lb. of power will balance

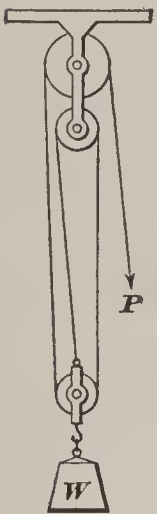


FIG. 7

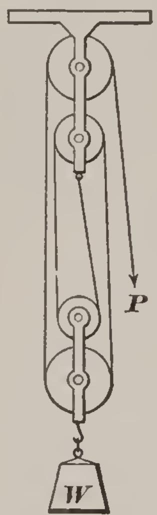


FIG. 8

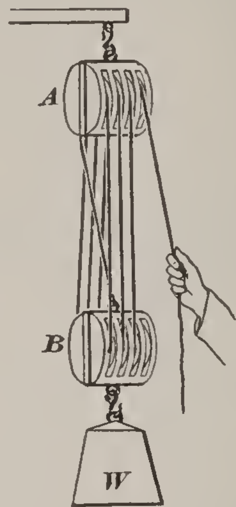


FIG. 9

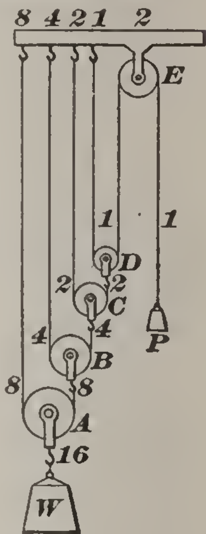


FIG. 10

3 lb. of weight. In the combination shown in Fig. 8, 1 lb. will sustain a weight W of 4 lb. but it must descend 4 in. to raise the weight 1 in. Fig. 9 represents the ordinary tackle block used by mechanics. The power applied for balance can be calculated by the following general rule:

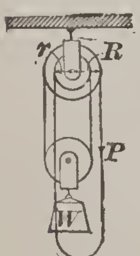


FIG. 11

Rule.—*In any combination of pulleys where one continuous rope is used, a load on the free end will balance a weight on the movable block as many times as great as the load on the free end as there are parts of the rope supporting the load, not counting the free end.*

In the combination shown in Fig. 10, each cord marked *1* has a tension equal to the power P; each cord marked *2*, has a tension equal to 2 times P; and so on with the other cords. As the sum of the tensions acting on the weight W is 16, $W=16P$. If n = number of pulleys, $P=\frac{W}{2^n}$; $W=2^nP$.

In the *differential pulley*, shown in Fig. 11, $W=\frac{2PR}{R-r}$

FRICTION AND LUBRICATION

Friction.—Friction is the resistance to motion due to the contact of surfaces; it is of two kinds, *sliding* and *rolling.* If the surface of a body could be made perfectly smooth, there would be no friction; but, in spite of the most perfect polish, the microscope reveals minute projections and cavities. As no surface can be made perfectly smooth, some separation of the two bodies must, in all cases, take place in order to clear such projections as exist on the surfaces. Therefore, friction is always more or less affected by the amount of the perpendicular pressure that tends to keep them together. The ultimate friction is the greatest frictional resistance that one body sliding over another is capable of opposing to any sliding force when at rest.

The *coefficient of friction* is the proportion that the ultimate friction in a given case bears to the perpendicular pressure. The coefficient of friction is usually expressed in decimals; but sometimes, as in the case of cars and engines, it is expressed in pounds (of friction) per ton. The coefficient of friction equals the ultimate friction divided by the perpendicular pressure, and the ultimate friction equals the perpendicular pressure multiplied by the coefficient of friction. Thus, if a block weighing 100 lb., stands on another block, and it takes a pressure of 35 lb. to slide it, the coefficient of friction $= \frac{35}{100}$, or .35.

Lubrication.—To diminish the friction, oil or grease is placed on the surfaces of sliding bodies so as to fill the cavities and spaces between the projections; this oil or grease is called a *lubricant.* There is probably no factor that has a more direct bearing on the cost of production per ton of coal and ores than the lubrication of mine machinery, and yet it is doubtful if there is another item connected with the operation of a mine less understood by owners, managers, and engineers in charge.

BEST LUBRICANTS FOR DIFFERENT PURPOSES

Low temperatures, as in rock drills driven by compressed air	Light mineral lubricating oils.

Very great pressures, slow speed	Graphite, soapstone, and other solid lubricants.
Heavy pressures, with slow speed	The above, and lard, tallow, and other greases.
Heavy pressures and high speed	Sperm oil, castor oil, and heavy mineral oils.
Light pressures and high speed	Sperm, refined petroleum, olive, rape, cottonseed.
Ordinary machinery	Lard oil, tallow oil, heavy mineral oils, and the heavier vegetable oils.
Steam cylinders	Heavy mineral oils, lard, tallow.
Watches and other delicate mechanism	Clarified sperm, neat's foot, porpoise, olive, and light mineral lubricating oils.

For mixture with mineral oils, sperm is best; lard is much used; olive and cottonseed are good.

STRENGTH OF MATERIALS

USEFUL FORMULAS

The ultimate strengths of different materials vary greatly from the average values given in the following tables. In actual practice, the safest procedure is to make a test of the material for its ultimate strength and coefficient of elasticity, or else specify in the contract that it shall not fall below certain prescribed limits. In the following formulas,

A = area of cross-section of material, in square inches;
E = coefficient of elasticity, in pounds per square inch;
G^2 = square of least radius of gyration;
I = moment of inertia about an axis passing through center of gravity of cross-section;
M = maximum bending moment, in inch-pounds;
P = total stress, in pounds;
R = moment of resistance;

S = ultimate stress, in pounds per square inch of area of section;
W = weight placed on a beam, in pounds;
b = breadth of cross-section of beam, in inches;
d = depth of beam (in.) = diam. of circ. section = altitude of triangular section = length of vertical side;
e = amount of elongation or shortening, in inches;
f = factor of safety;
l = length, in inches;
p = pressure, in pounds per square inch;
π = ratio of circumference to diameter = 3.1416, nearly;
q = constant used in formula for columns;
r = radius of circular section;
s = elastic set or deflection of a beam under a transverse (bending) stress, in inches;
t = thickness of shell or hollow section.

Tension, Compression, and Shear.—For tension, compression (where the piece does not exceed 10 times its least diameter) and shear,

$$P = \frac{AS}{f}$$

Breaking Stress.—To find the breaking stress, P, make $f = 1$. For safe load, take the values for f and S from the accompanying tables of Factors of Safety and Ultimate Strengths, espectively.

FACTORS OF SAFETY

Material	Steady Stress	Varying Stress	Shocks (Machines)
Cast iron..........	6	15	20
Wrought iron......	4	6	10
Steel..............	5	7	15
Wood.............	8	10	15
Brick and stone....	15	25	30

ULTIMATE STRENGTHS

Material	Tension	Compression	Shear	Flexure
Cast iron	20,000	90,000	20,000	36,000
Wrought iron	50,000	50,000	47,000	50,000
Steel	100,000	150,000	70,000	120,000
Wood	10,000	8,000	600 to 3,000	9,000
Stone		6,000		2,000
Brick	200	2,500		

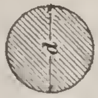
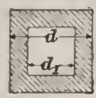
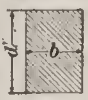
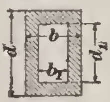
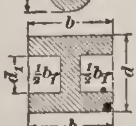
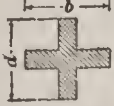
COEFFICIENT OF ELASTICITY

Material	Coefficient of Elasticity	Elastic Limit for Tension
Cast iron	15,000,000	6,000
Wrought iron	25,000,000	25,000
Steel	30,000,000	50,000
Wood	1,500,000	3,000

BENDING MOMENT AND DEFLECTION OF BEAMS

Kind of Beam and Manner of Loading	Bending Moment M	Deflection s
Cantilever, load at end	Wl	$\frac{1}{3}\frac{Wl^3}{EI}$
Cantilever, uniformly loaded	$\frac{1}{2}Wl$	$\frac{1}{8}\frac{Wl^3}{EI}$
Simple beam, load at middle	$\frac{1}{4}Wl$	$\frac{1}{48}\frac{Wl^3}{EI}$
Simple beam, uniformly loaded	$\frac{1}{8}Wl$	$\frac{5}{384}\frac{Wl^3}{EI}$
Beam fixed at both ends, load at middle	$\frac{1}{8}Wl$	$\frac{1}{192}\frac{Wl^3}{EI}$
Beam fixed at both ends, uniformly loaded	$\frac{1}{12}Wl$	$\frac{1}{384}\frac{Wl^3}{EI}$

PROPERTIES OF VARIOUS SECTIONS

Section		I	R	G^2
Solid circular...		$\frac{\pi d^4}{64}$	$\frac{\pi d^3}{32}$	$\frac{d^2}{16}$
Hollow circular...		$\frac{\pi(d^4-d_1^4)}{64}$	$\frac{\pi(d^4-d_1^4)}{32d}$	$\frac{d^2+d_1^2}{16}$
Solid square		$\frac{d^4}{12}$	$\frac{d^3}{6}$	$\frac{d^2}{12}$
Hollow square....		$\frac{d^4-d_1^4}{12}$	$\frac{d^4-d_1^4}{6d}$	$\frac{d^2+d_1^2}{12}$
Solid rectangular		$\frac{bd^3}{12}$	$\frac{bd^2}{6}$	$\frac{b^2}{12}$
Hollow rectangular		$\frac{bd^3-b_1d_1^3}{12}$	$\frac{bd^3-b_1d_1^3}{6d}$	$\frac{b^3d-b_1^3d_1}{12(bd-b_1d_1)}$
Solid triangular		$\frac{bd^3}{36}$	$\frac{db^2}{24}$	$\frac{d^2}{18}$
Solid elliptic ...		$\frac{\pi bd^3}{64}$	$\frac{\pi bd^2}{32}$	$\frac{b^2}{16}$
Hollow elliptic ...		$\frac{\pi}{64}(bd^3-b_1d_1^3)$	$\frac{\pi(bd^3-b_1d_1^3)}{32d}$	$\frac{b^3d-b_1^3d_1}{16(bd-b_1d_1)}$
I-beam		$\frac{bd^3-b_1d_1^3}{12}$	$\frac{bd^3-b_1d_1^3}{6d}$	$\frac{b^3d-b_1^3d_1}{12(bd-b_1d_1)}$
Cross with equal arms				$\frac{d^2}{22.5}$
Angle with equal arms				$\frac{d^2}{25}$

Elongation or Shortening Under Stress.—The amount of elongation or of shortening of a piece under a stress is given by the formula

$$e=\frac{Pl}{AE}$$

The coefficient of elasticity E must be taken from the accompanying table.

Breaking Strength of a Beam.—To find the breaking strength of a beam, use the formula

$$M=SR$$

Obtain M and R from the accompanying tables according to the kind of beam and nature of cross-section. A simple beam is one merely supported at its ends. In the expression for R, d is always understood to be the vertical side or depth; hence, that beam is the stronger that always has its greatest depth or longest side vertical. The moment of inertia I is taken about an axis perpendicular to d, and lying in the same plane.

The value of S for beams should be taken from the flexure column of table of Ultimate Strengths.

Deflection in Beams Due to Loads.—To find the amount of deflection in a beam due to a load, substitute the values of W, l, E, and I in the different expressions for the deflection s in the table Bending Moment and Deflection of Beams. The value of I is to be taken from the table Properties of Various Sections.

Columns.—To find the breaking strength of a column, use the formula:

$$P=\frac{SA}{1+q\frac{l^2}{G^2}}$$

The values of these quantities are taken from the accompanying tables.

Ropes and Chains.—Let D = diameter of rope, in inches = diameter of iron from which link in chain is made;

W = safe load, in tons of 2,000 lb.

For common hemp rope, $W=\frac{1}{3}D^2$.

For iron-wire rope, $W=\frac{8}{3}D^2$.
For steel-wire rope, $W=\frac{14}{3}D^2$.

CONSTANT USED IN FORMULA FOR COLUMNS

Material	Both Ends Flat or Fixed	One End Round	Both Ends Round
Cast iron..........	$\frac{1}{5,000}$	$\frac{1.78}{5,000}$	$\frac{4}{5,000}$
Wrought iron......	$\frac{1}{36,000}$	$\frac{1.78}{36,000}$	$\frac{4}{36,000}$
Steel..............	$\frac{1}{25,000}$	$\frac{1.78}{25,000}$	$\frac{4}{25,000}$
Wood.............	$\frac{1}{3,000}$	$\frac{1.78}{3,000}$	$\frac{4}{3,000}$

For close-link wrought-iron chain, $W=6D^2$.
For stud-link wrought-iron chain, $W=9D^2$.

WIRE ROPES

Wire ropes for mine use are made of either iron or steel, and are generally round. Flat wire ropes are sometimes employed, but the round rope is the one generally used in American practice, except in some of the deep metal mines having small compartment shafts. Steel ropes are in most respects superior to iron ropes, and are therefore gaining favor every year. Their principal advantage is their greater strength; consequently, they can be made lighter and can pass around pulleys and drums with less injury than an iron rope of equal strength.

Where great flexibility is required, such as in hoisting ropes, the strands are usually made up of 19 wires each, while haulage ropes have but 7 wires to the strand; yet, both kinds have 6 strands. A hemp core is generally used, and in some cases a core is also placed in each strand, to further increase the flexibility of the rope.

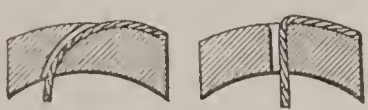
The *lay* of the rope is the twist or pitch of the wires in the strand, or of the strands in the rope. As the lay of the wires is less than that of the strands, each wire is exposed to external wear for short distances at intervals along the rope.

In the ordinary lay, Fig. 1 (*a*), the wires are twisted in the opposite direction to the strands; this method prevents the rope from untwisting when in use, and the wires from unraveling when they are worn through or broken at the surface.

In the Lang lay, view (*b*), the wires are twisted in the same direction as the strands, thus giving each wire a greater wearing surface, while the rope is smoother and will wear longer. After the wires begin to break, unraveling becomes troublesome, and it is more difficult to splice a Lang-lay rope than an ordinary-lay one. Hoisting ropes, especially those used to raise and lower men, should not be spliced.


(*a*) (*b*) (*c*)
FIG. 1

The locked-wire rope, a cross-section of which is shown in (*c*), consists of wires of special cross-section formed in concentric layers. The lay of the inner wires is opposite to that of the outer ones, and somewhat longer.


(*a*) (*b*)
FIG. 2

In fastening a rope to a drum, a great error is often made. Men who would not think of passing a rope around a pulley of too small diameter will insert it in the drum rim in such a way as to make a very sharp curve, as shown in Fig. 2 (*a*), and make a weak point in the rope that would not otherwise exist. The right way of passing the rope through the drum rim is shown in (*b*).

Flattened-Strand Ropes.—Many ropes have flattened strands, as shown in Fig. 3; several wires thus take the wear of the rope instead of a single one, as is the case with a round strand when new. The manufacturers claim for these ropes longer life, more

uniform wear, greater flexibility, less liability of wires becoming brittle, and freedom from all tendency to spin or kink. It

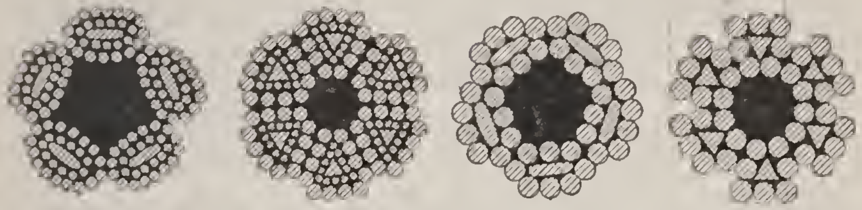


Fig. 3

is also claimed that the smoother surface effects considerable saving in the wear of pulleys and sheaves.

STRESS IN HOISTING ROPES ON INCLINED PLANES

Rise per 100 Ft. Horizontal Feet	Angle of Inclination	Stress per Ton of 2,000 Lb. Pounds	Rise per 100 Ft. Horizontal Feet	Angle of Inclination	Stress per Ton of 2,000 Lb. Pounds
5	2° 52′	140	105	46° 24′	1,484
10	5° 43′	240	110	47° 44′	1,516
15	8° 32′	336	115	49° 00′	1.535
20	11° 10′	432	120	50° 12′	1,573
25	14° 03′	527	125	51° 21′	1,597
30	16° 42′	613	130	52° 26′	1.620
35	19° 18′	700	135	53° 29′	1,642
40	21° 49′	782	140	54° 28′	1,663
45	24° 14′	860	145	55° 25′	1,682
50	26° 34′	933	150	56° 19′	1,699
55	28° 49′	1,003	155	57° 11′	1,715
60	30° 58′	1,067	160	58° 00′	1,730
65	33° 02′	1,128	165	58° 47′	1,744
70	35° 00′	1,185	170	59° 33′	1,758
75	36° 53′	1,238	175	60° 16′	1,771
80	38° 40′	1,287	180	60° 57′	1,782
85	40° 22′	1,332	185	61° 37′	1,794
90	42° 00′	1,375	190	62° 15′	1,804
95	43° 32′	1,415	195	62° 52′	1,813
100	45° 00′	1,450	200	63° 27′	1,822

Wire-Rope Tables.—The accompanying wire-rope table is a rearrangement of the standard tables published in the

HOISTING, HAULAGE, AND TRANSMISSION ROPES

Use of Rope	Diameter Inches	Approximate Circumference. Inches	Weight per Linear Foot Pounds	Swedish Iron			Crucible Cast Steel			Extra Strong Crucible Cast Steel			Plow Steel		
				Approximate Breaking Strength Tons	Proper Working Load. Tons	Proper Size of Drum or Sheave. Feet	Approximate Breaking Strength Tons	Proper Working Load. Tons	Proper Size of Drum or sheave. Feet	Approximate Breaking Strength Tons	Proper Working Load. Tons	Proper Size of Drum or Sheave. Feet	Approximate Breaking Strength Tons	Proper Working Load. Tons	Proper Size of Drum or Sheave. Feet
Hoisting ropes, 6-strand, 19-wire, hemp center	$2\frac{3}{4}$	$8\frac{5}{8}$	11.950	114.0	22.80	15	228.0	45.60	11	266.00	53.00	11	305.00	61.00	15
	$2\frac{1}{2}$	$7\frac{7}{8}$	9.850	95.0	18.90	$13\frac{1}{2}$	190.0	37.90	10	222.00	45.00	10	254.00	50.80	$13\frac{1}{2}$
	$2\frac{1}{4}$	$7\frac{1}{8}$	8.000	78.0	15.60	12	156.0	31.20	9	182.00	36.40	9	208.00	41.60	12
	2	$6\frac{1}{4}$	6.300	62.0	12.40	11	124.0	24.80	8	144.00	28.80	8	165.00	33.00	11
	$1\frac{3}{4}$	$5\frac{1}{2}$	4.850	48.0	9.60	9	96.0	19.20	7	112.00	22.40	$7\frac{1}{2}$	128.00	25.60	9
	$1\frac{5}{8}$	5	4.150	42.0	8.40	$8\frac{1}{2}$	84.0	16.80	$6\frac{1}{2}$	97.00	19.40	$6\frac{1}{2}$	111.00	22.20	$8\frac{1}{2}$
	$1\frac{1}{2}$	$4\frac{3}{4}$	3.550	36.0	7.20	8	72.0	14.40	6	84.00	16.80	6	96.00	19.20	8
	$1\frac{3}{8}$	$4\frac{1}{4}$	3.000	31.0	6.20	$7\frac{1}{2}$	62.0	12.40	$5\frac{1}{2}$	72.00	14.40	$5\frac{1}{2}$	82.00	16.40	$7\frac{1}{2}$
	$1\frac{1}{4}$	4	2.450	25.0	5.00	7	50.0	10.00	5	58.00	11.60	5	67.00	13.40	7

Hoisting ropes, 6 strand, 19-wire, hemp-center	1⅛	3½	2.000	21.0	4.20	6	42.0	8.40	4½	49.00	9.80	4½	56.00	11.20	6
	1	3	1.580	17.0	3.40	5	34.0	6.80	4	39.00	7.80	4	44.00	8.80	5
	7/8	2¾	1.200	13.0	2.60	4½	26.0	5.20	3½	30.00	6.00	3½	34.00	6.80	4½
	3/4	2¼	.890	9.7	1.94	4	19.4	3.88	3	22.00	4.40	3	25.00	5.00	4
	5/8	2	.620	6.8	1.36	3½	13.6	2.72	2½	15.80	3.16	2½	18.00	3.60	3½
	9/16	1¾	.500	5.5	1.10	3	11.0	2.20	2¼	12.70	2.54	2¼	14.50	2.90	3
	1/2	1½	.390	4.4	.88	2¾	8.8	1.76	2	10.10	2.02	2	11.40	2.28	2¾
	7/16	1¼	.300	3.4	.68	2½	6.8	1.36	1¾	7.80	1.56	1¾	8.85	1.77	2½
	3/8	1⅛	.220	2.5	.50	2	5.0	1.00	1½	5.78	1.15	1½	6.55	1.31	2
	5/16	1	.150	1.7	.34	1½	3.4	.68	1	4.05	.81	1	4.50	.90	1½
	1/4	¾	.100	1.2	24	¾	2.4	.48	½	2.70	.54	½	3.00	.60	¾
Haulage and transmission ropes, 6-strand, 7-wire, hemp center	1½	4¾	3.550	34.0	6.80	11	68.0	13.60	8	79.00	15.80	8	91.00	18.20	11
	1⅜	4¼	3.000	29.0	5.80	10	58.0	11.60	7½	68.00	13.60	7½	78.00	15.60	10
	1¼	4	2.450	24.0	4.80	9¼	48.0	9.60	7	56.00	11.20	7	64.00	12.80	9¼
	1⅛	3½	2.000	20.0	4.00	8	40.0	8.00	6	46.00	9.20	6	53.00	10.60	8
	1	3	1.580	16.0	3.20	6¾	32.0	6.40	5	37.00	7.40	5	42.00	8.40	6¾
	7/8	2¾	1.200	12.0	2.40	6	24.0	4.80	4½	28.00	5.60	4½	32.00	6.40	6
	3/4	2¼	.890	9.3	1.86	5¼	18.6	3.72	4	21.00	4.20	4	24.00	4.80	5¼
	11/16	2⅛	.750	7.9	1.58	4¾	15.8	3.16	3¾	18.40	3.68	3¾	21.00	4.20	4¾
	5/8	2	.620	6.6	1.32	4½	13.2	2.64	3½	15.10	3.02	3½	17.00	3.40	4½
	9/16	1¾	.500	5.3	1.06	4	10.6	2.12	3	12.30	2.46	3	14.00	2.80	4
	1/2	1½	.390	4.2	.84	3¾	8.4	1.68	2¾	9.70	1.94	2¾	11.00	2.20	3¾
	7/16	1¼	.300	3.3	.66	3½	6.6	1.32	2½	7.50	1.50	2½	8.55	1.71	3½
	3/8	1⅛	.220	2.4	.48	2¾	4.8	.96	2	5.58	1.11	2	6.35	1.27	2¾
	5/16	1	.150	1.7	.34	2	3.4	.68	1½	3.88	.77	1½	4.35	.87	2
	9/32	7/8	.125	1.4	.28	1¾	2.8	.56	1¼	3.22	.64	1¼	3.65	.73	1¾

catalogs of most of the American manufacturers of wire rope. The proper working load given in this table is one-fifth of the approximate breaking stress, that is, a factor of safety of 5 is used, and when the values given in this table are used this factor is supposed to allow for the bending stress. The sizes of sheaves or drums given in this table are largely empirical, but they are based on a long experience in the use of wire ropes, and in most cases represent the minimum diameter recommended by the rope makers. The factor of safety of 5 assumes ordinary conditions of working; where the conditions are extraordinary, and particularly in cases where men are to be hoisted, a larger factor than 5 is used, varying from 5 to 10.

Stress in Hoisting Ropes on Inclined Planes.—The accompanying table is based upon an allowance of 40 lb. per T. for rolling friction, but there will be an additional stress due to the weight of the rope and inclination of the plane.

STARTING STRAIN ON HOISTING ROPES

Experiment	Strain in Rope Pounds
Empty cage, lifted gently....................	4,030
Empty cage, started with 2½ in. of slack rope...	5,600
Empty cage, started with 6 in. of slack rope....	8,950
Empty cage, started with 12 in. of slack rope...	12,300
Cage and loaded cars, as weighed..............	11,300
Cage, and loaded cars, lifted slowly and gently..	11,525
Cage and loaded cars, started with 3 in. of slack rope....................................	19,025
Cage and loaded cars, started with 6 in. of slack rope....................................	24,625
Cage and loaded cars, started with 9 in. of slack rope....................................	26,850

Starting Strain on Hoisting Rope.—In selecting a hoisting rope, due allowance must be made for the shock and extra strain imposed on the rope when the load is started from rest. Experiments made by placing a dynamometer between the rope and the cage have shown that the starting stress may be from two to three times the actual load.

The following table shows the results of a number of tests under different conditions, with slack chain, amount of load, and speed in starting.

EXTRA STRAIN ON A HOISTING ROPE WITH A FEW INCHES OF SLACK CHAIN

Dynamometer Tests	Tons	Hundred-weights
First Test		
Empty cage lifted gently	1	16
Empty cage with $2\frac{1}{2}$ in. slack chain	2	10
Empty cage with 6 in. slack chain	4	0
Empty cage with 12 in. slack chain	5	10
Second Test		
Cage and 4 empty cars weighed by machine	2	17
Cage and 4 empty cars lifted gently	3	0
Cage and 4 empty cars with 3 in. slack chain	5	0
Cage and 4 empty cars with 6 in. slack chain	5	10
Cage and 4 empty cars with 12 in. slack chain	7	0
Third Set		
Cage and full cars weighed by machine	5	1
No. 1, lifted gently	5	1
No. 2, lifted gently	5	3
No. 1, with 3 in. slack chain	8	10
No. 2, with 3 in. slack chain	8	10
No. 1, with 6 in. slack chain	10	10
No. 2, with 6 in. slack chain	11	10
No. 1, with 9 in. slack chain	12	10
No. 2, with 9 in. slack chain	11	10

Sheaves.—To decrease the bending stresses the sheaves for wire-rope transmissions are generally of as large diameter as is practicable to give the required speed to the rope. Large sheaves are also advantageous because with them the rope is run at a high velocity allowing of a lower tension and permitting a rope of smaller diameter to be used than would be possible with smaller sheaves, provided, of course, that the span is of sufficient length to give the necessary weight.

Sheaves are generally made of cast iron when not exceeding 12 ft. in diameter, and when larger than this they are usually built up with wrought-iron arms. Sheaves, upon which the rope is to make but a single half-turn, are made with **V**-shaped grooves in their circumference. The bottom part of the groove is widened to receive the filling, which consists of some substance to give a bed for the rope to run on and protect it from wear, and to increase the friction so that the rope will not slip. This filling is made of blocks of wood, rubber, leather, or other material. Rubber and leather have been used separately, but blocks of rubber separated by pieces of leather have been found to give the best rcsults.

In the accompanying table, the ropes are made of cast steel and used on inclines. The ropes are composed of 6 strands of 7 wires each and have hemp cores.

EFFECTS OF VARIOUS SIZE SHEAVES OR DRUMS ON LIFE OF WIRE ROPES

Diameter of Rope Inches	Percentages of Life for Various Diameters						
	100	90	80	75	60	50	25
	Diameter of Sheaves or Drums, in Feet						
$1\frac{1}{2}$	16.00	14.00	12.00	11.00	9.00	7.00	4.75
$1\frac{3}{8}$	14.00	12.00	10.00	8.50	7.00	6.00	4.50
$1\frac{1}{4}$	12.00	10.00	8.00	7.25	6.00	5.50	4.25
$1\frac{1}{8}$	10.00	8.50	7.75	7.00	6.00	5.00	4.00
1	8.50	7.75	6.75	6.00	5.00	4.50	3.75
$\frac{7}{8}$	7.75	7.00	6.25	5.75	4.50	3.75	3.25
$\frac{3}{4}$	7.00	6.25	5.50	5.00	4.25	3.50	2.75
$\frac{5}{8}$	6.00	5.25	4.50	4.00	3.25	3.00	2.50
$\frac{1}{2}$	5.00	4.50	4.00	3.50	2.75	2.25	1.75

Wire-Rope Calculations.—The *working load*, also called the proper working load, is the maximum load that a rope should be permitted to support under working conditions. When a load is attached, the stress on a rope bending over a sheave, is

WIRE AND SHEET-METAL GAUGES

Gauge Number	U. S. Standard Sheet-Metal Gauge Inch	British Imperial Standard Wire Gauge Mm.	Birmingham Gauge Inch	American or Brown & Sharpe Gauge Inch	Roebling's Gauge Inch	Trenton Wire Co.'s Wire Gauge Inch
0000000	.5	12.7			.49	
000000	.469	11.78			.46	
00000	.438	10.97			.43	.45
0000	.406	10.16	.454	.46	.393	.40
000	.375	9.45	.425	.40964	.362	.36
00	.344	8.84	.38	.3648	.331	.33
0	.313	8.23	.34	.32486	.307	.305
1	.281	7.62	.3	.2893	.283	.285
2	.266	7.01	.284	.25763	.263	.265
3	.25	6.4	.259	.22942	.244	.245
4	.234	5.89	.238	.20431	.225	.225
5	.219	5.38	.22	.18194	.207	.205
6	.203	4.88	.203	.16202	.192	.19
7	.188	4.47	.18	.14428	.177	.175
8	.172	4.06	.165	.12849	.162	.16
9	.156	3.66	.148	.11443	.148	.145
10	.141	3.26	.134	.10189	.135	.13
11	.125	2.95	.12	.09074	.12	.1175
12	.109	2.64	.109	.08081	.105	.105
13	.094	2.34	.095	.07196	.092	.0925
14	.078	2.03	.083	.06408	.08	.08
15	.07	1.83	.072	.05707	.072	.07
16	.0625	1.63	.065	.05082	.063	.061
17	.0563	1.42	.058	.04526	.054	.0525
18	.05	1.22	.049	.0403	.047	.045
19	.0438	1.01	.042	.03589	.041	.04
20	.0375	.91	.035	.03196	.035	.035
22	.0313	.71	.028	.02535	.028	.028
24	.025	.56	.022	.0201	.023	.0225
26	.0188	.45	.018	.01594	.018	.018
28	.0156	.38	.014	.01264	.016	.016
30	.0125	.31	.012	.01002	.014	.014
32	.0101	.27	.009	.00795	.013	.012
34	.0086	.23	.007	.0063	.01	.01
36	.007	.19	.004	.005	.009	.009
38	.0063	.15		.00396	.008	.008
40		.12		.00314	.007	.007

made up of two parts: (1) That due to the load on the rope, known as the *load stress*; (2) that due to the bending of the rope about a sheave or drum, known as the *bending stress*. That is, if S is the total safe stress, S_b is the bending stress, S_l is the load stress, $S=S_b+S_l$ and $S_l=S-S_b$. The total stress must not equal the elastic limit of the material composing the rope and is usually taken as from one-third to one-fourth the approximate breaking stress. The proper size of rope to use to hoist a given weight may be taken directly from the tables just given, but these tables do not take account of the bending stress, except by allowing for it in the factor of safety assumed.

The general formula for the bending stress is

$$S_b=\frac{EaA}{D},$$

in which S_b = bending stress;
E = modulus of elasticity;
a = diameter of each wire;
D = diameter of drum or sheave;
A = total area of wire cross-section, in inches.

Wear of Wire Ropes.—The deterioration of wire ropes may be either external or internal, and may be due (1) to abrasion, due to the rubbing of the outside surface of the rope against other objects, or to the internal chafing of the wires composing the strands against one another; (2) to injury from overloading, to shock due to sudden starting of the load, or to repeated bendings about too sharp angles or over sheaves or rollers of too small a diameter for the size of the rope; (3) to rust or corrosion of the wire from acid waters, or to decay of the hemp cores.

Lubrication of Ropes.—Mine water has a very corrosive action on wire ropes, and a rope will soon be destroyed unless the water is prevented from coming in contact with the metal of the rope. To avoid this corrosive action tar, black oil, or some lubricating preparation is applied to the rope, but any lubricant used must be free from acids or other substances that would corrode the wire. The lubricants to be used are generally specified by the manufacturers of the ropes, a list of which can be obtained from them when purchasing a rope.

Wire-Rope Fastenings.—An ordinary form of *thimble-spliced fastening* is shown in Fig. 1 (*a*). In this method, the wires, after being frayed out at the end and the rope bent around the thimble, are laid snugly about the main portion of the rope and securely fastened by wrapping with stout wire; the extreme ends that project below this wrapping are then folded back, as shown. Another style of thimble splicing is shown in (*b*). In this case the strands are interlocked as in splicing, and the joint is wrapped with wire as in the former method.

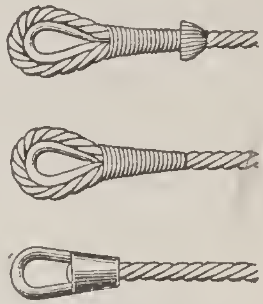


FIG. 1

The *socket fastening* is shown in (*c*); the hole in which the rope end is fastened is conical in shape. The rope is generally secured by fraying out the wires at the end, the interstices being filled up with spikes driven in tightly. The whole is finally cemented by pouring in molten Babbitt metal.

A good fastening can be made if the wires, after being frayed out at the end, are bent upon themselves in hook fashion, the prongs of some being longer than others, so that the bunch will conform to the conical aperture of a socket, and the melted Babbitt metal finally run in as usual. This makes a much neater fastening than either of those shown in (*a*) and (*b*), but it does not possess nearly as much strength. The thimble possesses a serious disadvantage; it is usually made of a piece of curved metal bent around into an oval shape, as shown in (*a*) and (*b*), with the groove, in which the rope lies, outside, the ends coming together n a sharp point. When weight is placed on the rope, the strain on the thimble is apt to cause one end to wedge itself beyond or past the other, and with its sharp edge cut the strands in the splice.

Splicing a Wire Rope.—To splice a wire rope the only tools needed are a cold cutter and hammer for cutting and trimming the strands, and two needles 12 in. long, made of good steel and tapered ovally to a point. Cut off the ends of the ropes to be spliced and unlay three adjacent strands of each back 15 ft.; cut out the hemp center to this point and relay the strands for 7 ft. and cut them off. Pull the ropes by each other until they

have the position shown in Fig. 2 (*a*), cut off *a* and *d′*, *b* and *c′*, view (*b*), making their lengths approximately 10 and $12\frac{1}{2}$ ft., respectively, measured from the point where the hemp centers are cut. Place the ropes together, view (*b*); unlay *e*, *d*, *c*, view (*a*), keeping the strands together, and follow with *e′*, *d′*, *c′*, view (*b*). Similarly, unlay *f′*, *a′*, *b′*, view (*b*), and follow with *f*, *a*, *b*, until the rope appears as in view (*c*). Next run the strands into the middle of the rope. To do this, cut off the end of the strand *e′*, view (*c*), so that when it is put in place it will just reach to the end *x* of the hemp core, and then push

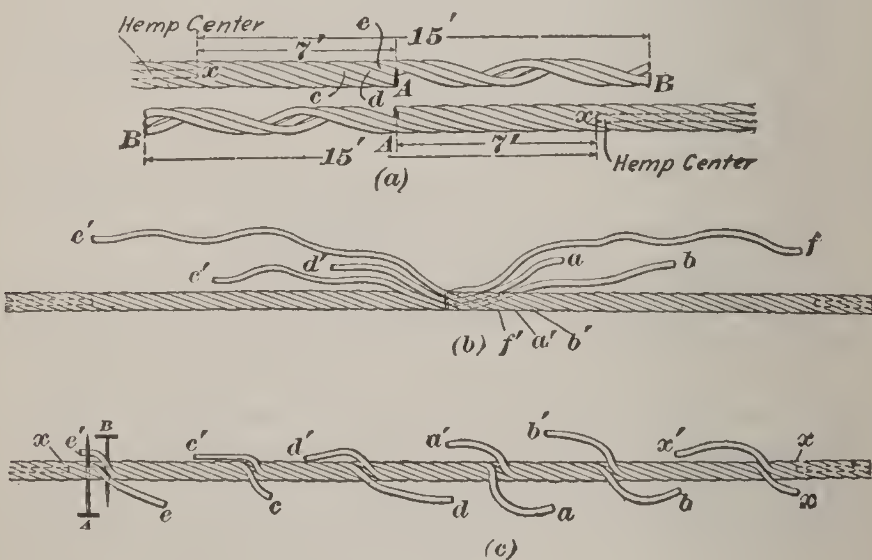


FIG. 2

one needle *A* through the rope from the under side, leaving two strands at the front of the needle, as shown. Push the other needle *B* through from the upper side and as close to the first needle *A* as possible, leaving the strands *e* and *e′* between them; place the first needle *A* on the knee and turn the other needle *B* around with the coil of the rope, and force the strand *e′* into the center of the rope. Repeat this operation with the other ends and cut them off so that the ends coming together in the center of the rope will butt against each other as nearly as possible.

Ordinary Long Splice.—The tools required to make a long splice in wire rope are a pair wire nippers, for cutting off strands; a pair pliers, for pulling through and straightening ends of strands; two marline-spikes, one round and one oval, for opening strands; a knife to cut hemp center; two clamps, to untwist rope to insert ends of strands, or, in place of them, two short hemp-rope slings, with a stick for each as a lever; a wooden mallet, and some rope twine. Also, a bench and vise are handy. The length of the splice depends on the size of the rope. The larger ropes require the longer splices. The splice of ropes from $\frac{5}{8}$ in. to $\frac{7}{8}$ in. in diameter should not be less than 20 ft.; from $\frac{7}{8}$ in. to $1\frac{1}{8}$ in., 30 ft.; and from $1\frac{1}{8}$ in. up, 40 ft.

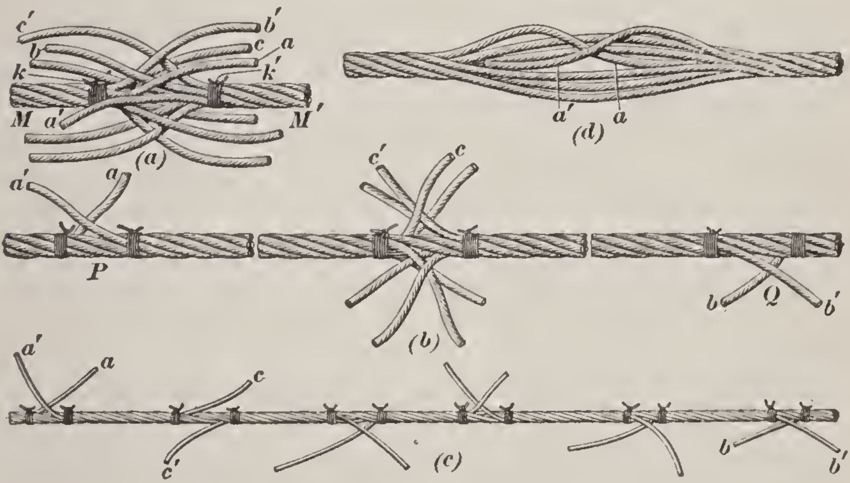


FIG. 3

To splice a rope, tie each end with a piece of cord at a distance equal to one-half the length of the splice, or 10 ft. back from the end, for a $\frac{5}{8}$ in. rope, after which unlay each end as far as the cord. Then cut out the hemp center, and bring the two ends together as close as possible, placing the strands of the one end between those of the other, as shown in Fig. 3 (*a*). Remove the cord *k* from one end *M* of the rope, and unlay any strand *a*, and follow it up with the strand of the other end *M′* of the rope that corresponds to it, as *a′*. Leave out about 6 in. of *a* and cut off *a′* about 6 in. from the rope, thus leaving two short ends, as shown at *P* in view (*b*), which must be tied for the present

by cords as shown. The cord k should again be wound around one end M of the rope, view (a), to prevent the unraveling of the strands. Remove the cord k' on the other end M' of the rope, and unlay the strand b; follow it up with the strand b', leaving out the ends and tying them down, as at view (b). Replace the cord k' for the same purpose as stated before. Again remove the cord k and unlay the next strand c, view (a), and follow it up with c', stopping, however, this time within 4 ft. of the first set. Continue this operation with the remaining 6 strands, stopping 4 ft. short of the preceding set each time. The strands are now in their proper places, with the ends passing each other at intervals of 4 ft., as shown in view (c). To dispose of the loose ends, clamp the rope in a vise at the left of the strands a and a', and fasten a clamp to the rope at the right of these strands; then remove the cords tied around the rope that holds these two strands down; after which turn the clamp in the opposite direction to which the rope is twisted, thereby untwisting the rope, as shown in view (d). The rope should be untwisted enough to allow its hemp core to be pulled out with a pair of nippers. Cut off 24 in. of the hemp core, 12 in. at each side from the point of intersection of the strands a and a', and push the ends of the strands in their place, as shown. Then allow the rope to twist up to its natural shape, and remove the clamps. After the rope has been allowed to twist up, the strands tucked in generally bulge out somewhat. This bulging may be reduced by lightly tapping the bulged part of the strands with a wooden mallet, which will force their ends farther into the rope.

CHAINS

The links of iron chains are usually made as short as is consistent with easy play, so as to make them less liable to kink, and also to prevent bending when wound around drums, sheaves, etc. The weight of close-link chain is about 3 times the weight of the bar from which it is made, for equal lengths.

The strength of a chain link is less than twice that of a straight bar of a sectional area equal to that of one side of the link. A weld exists at one end and a bend at the other, each requiring at least one heat, which produces a decrease in the strength.

The report of the committee of the U. S. Testing Board, on tests of wrought-iron and chain cables, is shown in the accompanying table.

ULTIMATE RESISTANCE AND PROOF TESTS OF WROUGHT-IRON CHAIN CABLES

Diam. of Bar Inches	Average Resist. = 163% of Bar Pounds	Proof Test Pounds	Diam. of Bar Inches	Average Resist. = 163% of Bar Pounds	Proof Test Pounds
1	71,172	33,840	$1\frac{9}{16}$	162,283	77,159
$1\frac{1}{16}$	79,544	37,820	$1\frac{5}{8}$	174,475	82,956
$1\frac{1}{8}$	88,445	42,053	$1\frac{11}{16}$	187,075	88,947
$1\frac{3}{16}$	97,731	46,468	$1\frac{3}{4}$	200,074	95,128
$1\frac{1}{4}$	107,440	51,084	$1\frac{13}{16}$	213,475	101,499
$1\frac{5}{16}$	117,577	55,903	$1\frac{7}{8}$	227,271	108,058
$1\frac{3}{8}$	128,129	60,920	$1\frac{15}{16}$	241,463	114,806
$1\frac{7}{16}$	139,103	66,138	2	256,040	121,737
$1\frac{1}{2}$	150,485	71,550			

HORSEPOWER OF MANILA ROPES

Diam. of Rope. In.	Weight per Foot. Lb.	Breaking Strain Lb.	Working Strain. Lb.	1,000 Ft. per Min.		2,000 Ft. per Min.		3,000 Ft. per Min.		5,000 Ft. per Min.	
				H. P.	Tens. Wt.	H. P.	Tens. Wt.	H. P.	Tens. Wt.	H. P.	Tens. Wt.
$\frac{5}{8}$	.15	4,000	121	$2\frac{1}{4}$	90	$4\frac{1}{2}$	90	$6\frac{1}{4}$	80	$8\frac{1}{2}$	70
$\frac{3}{4}$	.18	5,000	151	$2\frac{3}{4}$	110	$5\frac{1}{2}$	110	$7\frac{3}{4}$	100	$10\frac{3}{4}$	90
$\frac{7}{8}$	.27	7,500	227	$4\frac{1}{4}$	170	$8\frac{1}{4}$	170	$11\frac{3}{4}$	160	16	130
1	.33	9,000	272	5	200	10	200	14	180	19	150
$1\frac{1}{8}$	.45	12,250	371	7	280	$13\frac{1}{2}$	270	19	250	26	210
$1\frac{1}{4}$	.50	14,000	424	8	320	$15\frac{1}{2}$	310	22	290	$29\frac{1}{2}$	240
$1\frac{3}{8}$	.65	18.062	547	$10\frac{1}{4}$	410	20	400	$28\frac{1}{4}$	370	$38\frac{1}{2}$	310
$1\frac{1}{2}$	.73	20,250	613	$11\frac{1}{2}$	460	22	440	$31\frac{1}{2}$	420	$43\frac{1}{2}$	350
$1\frac{5}{8}$	.82	25,000	760	$14\frac{1}{4}$	570	$27\frac{3}{4}$	550	$39\frac{1}{2}$	520	$55\frac{1}{2}$	448
$1\frac{3}{4}$	1.08	30,250	916	17	680	$33\frac{1}{2}$	660	$47\frac{1}{4}$	630	$64\frac{3}{4}$	520
2	1.27	36,000	1,000	$20\frac{1}{2}$	810	40	790	$56\frac{1}{2}$	740	$77\frac{1}{2}$	620

HYDROMECHANICS

HYDROSTATICS

Hydrostatics treats of liquids at rest under the action of forces. If a liquid is acted on by a pressure, the pressure per unit of area exerted anywhere on the mass of liquid is transmitted undiminished in all directions, and acts with the same force on all surfaces, in a direction at right angles to those surfaces.

Downward Pressure of Liquids.—The pressure on the bottom of a vessel containing a liquid is independent of the shape of the vessel, and is equal to the weight of a prism of the liquid whose base is the same as the bottom of the vessel, and whose altitude is the distance between the bottom and the upper surface of the liquid, plus the pressure per unit of area upon the upper surface of the liquid multiplied by the area of the bottom of the vessel.

Upward Pressure of Liquids.—The upward pressure on any submerged horizontal surface equals the weight of a prism of the liquid whose base has an area equal to the area of the submerged surface, and whose altitude is the distance between the submerged surface and the upper surface of the liquid, plus the pressure per unit of area on the upper surface of the liquid multiplied by the area of the submerged surface.

Lateral Pressure of Liquids.—The pressure on any vertical surface due to the weight of the liquid is equal to the weight of a prism of the liquid whose base has the same area as the vertical surface, and whose altitude is the depth of the center of gravity of the vertical surface below the level of the liquid. Any additional pressure is to be added, as in the previous cases.

Pressure of Liquids on Oblique Surfaces.—The pressure exerted by a liquid in any direction on a plane surface is equal to the weight of a prism of the liquid whose base is the projection of the surface at right angles to the given direction, and whose height is the depth of the center of gravity of the surface below the level of the liquid.

If a cylinder is filled with water, and a pressure applied, the total pressure on any half section of the cylinder is equal to the projected area of the half cylinder (or diameter length of cylinder) multiplied by the depth of the center of gravity of the half cylinder, multiplied by the weight of 1 cu. in. of water, plus the diameter of the shell, multiplied by the pressure per square inch, multiplied by the length of the cylinder.

If d = diameter, in inches, and l = length of cylinder, in inches, the pressure due to the weight of the water when the cylinder is vertical upon the half cylinder $= d \times l \times \frac{l}{2} \times$ weight of 1 cu. in. of water $= d \times \frac{l^2}{2} \times$ weight of 1 cu. in. of water.

The *pressure*, in pounds per square inch, due to a head of water is equal to the head in feet multiplied by .434. The *head* equals the pressure, in pounds per square inch, multiplied by 2.304.

Flow of Water Through Pipes.—The following formulas for the flow of water through pipes are those arranged by Gould, in which

Q = amount of water, in cubic feet per second;
q = U. S. gallons per minute;
D = diameter of pipe, in feet;
d = diameter of pipe, in inches;
H = total head, in feet;
h = head per 1,000 ft.;
V = velocity, in feet per second.

Pipes above 8 in. in diameter, rough inside surface.

$$Q = \sqrt{D^5 h} = D^2 \sqrt{Dh}; \quad V = 1.27 \sqrt{Dh}$$

For diameter, in inches,

$$Q = \frac{d^2}{288} \sqrt{\frac{dh}{3}}$$

Pipes between 3 and 8 in. in diameter, rough inside surface,

$$Q = 0.89 \sqrt{D^5 h} = 0.89\ D^2 \sqrt{Dh}; \quad V = 1.13 \sqrt{Dh}$$

Large pipes, smooth inside surface,

$$Q = 1.4 \sqrt{D^5 h} = 1.4\ D^2 \sqrt{Dh}; \quad V = 1.78 \sqrt{Dh}$$

Small pipes, smooth inside surface,

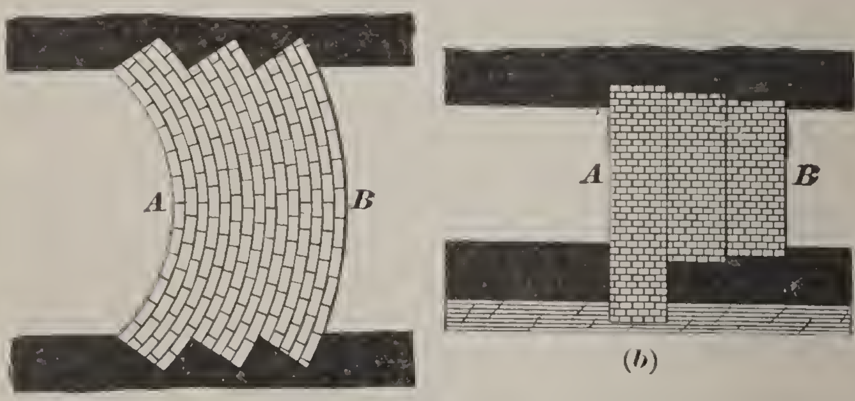
$$Q = 0.89 \sqrt{2 D^5 h} = 1.25\ D^2 \sqrt{Dh}; \quad V = 1.6 \sqrt{Dh}$$

It is best to calculate any pipe line by the formula for pipes having a rough internal surface, because all pipes become more or less rough with use.

Siphons.—When any part of the pipe line rises above the source of supply, such a line is called a *siphon.* If this rise is greater than the height of the water barometer (34 ft. at sea level), water will not flow through the siphon. The flow through the siphon will be the same as that through any pipe line so long as there is no accumulation of air at the highest point of the line; but such an accumulation will decrease or entirely stop the flow. All siphons should be provided at their highest points with valves for discharging the air and introducing water to fill the siphon, and it is usually best to trap the lower end of the pipe so that air cannot enter it, and to enlarge the upper end so as to reduce the loss of the stream in entering. For a siphon to work well, the fall between the intake and the discharge end should be considerable, if the rise amounts to much.

DAMS

Construction of Dams in Mines.—Dams may be constructed in mines, either to isolate a portion of the workings so that it



(a) Fig. 1

can be flooded to extinguish fires, or, where an extremely wet formation has been penetrated, to prevent the water from

flowing into the workings. But in either case the dam should have sufficient strength to resist any column of water that will come against it. The dam should be arched toward the direction from which the pressure comes, and should be given a good firm bearing in both walls and in the floor and roof. A brick dam that was constructed to isolate a portion of the seam so that it might be flooded to extinguish a mine fire is shown in Fig. 1, (*a*) being the plan and (*b*) a cross-section. This dam is composed of three brick arches, each 5 ft. thick, placed one against the other so that they act as one solid structure. The gangway at this point is about 20 ft. wide, and the distance to the next upper level is about 119 ft. It was intended that this should be the maximum head of water that the dam would have to resist, though it was made sufficiently strong to resist a head of water reaching the surface. The separate walls were constructed one at a time, and the cement allowed to set before the next wall was placed. The back wall was carried to a greater depth and height than the others, so as to make sure that all slips or partings had been closed. The total pressure upon the dam when the water was in the mine was about 70,000 lb. per sq. ft.

Dams constructed to permit the flooding of a mine usually require no passages through them, but dams constructed to confine the water to certain parts of the workings, and so reduce pumping charges, usually have both manways and drain pipes through them. Fig. 2 (*a*) shows the plan and (*b*) shows the cross-section of such a dam constructed to keep the water that came from some exploring drifts out of the mine workings. As originally constructed, it consisted of a sandstone dam 10 ft. thick and arched on the back face with a radius of 6 ft. A piece of 20-in. pipe provided a manway through the masonry and was held in place by three sets of clamps and bolts passing through the stone work. A 5-in. drain pipe was also carried through the dam and secured by clamps. When the pressure came upon the dam it was found to leak, so the water was drained off and a 22-in. brick wall built 2 ft. 4 in. back of the dam, the space between being filled with concrete, and the manway and drain pipe extended through the brick wall. Before closing the drain pipe, horse manure was fastened

against the face of the brick wall by means of a plank partition. After this the manway and drain pipe were closed, and when the pressure came on, the dam was found to leak a

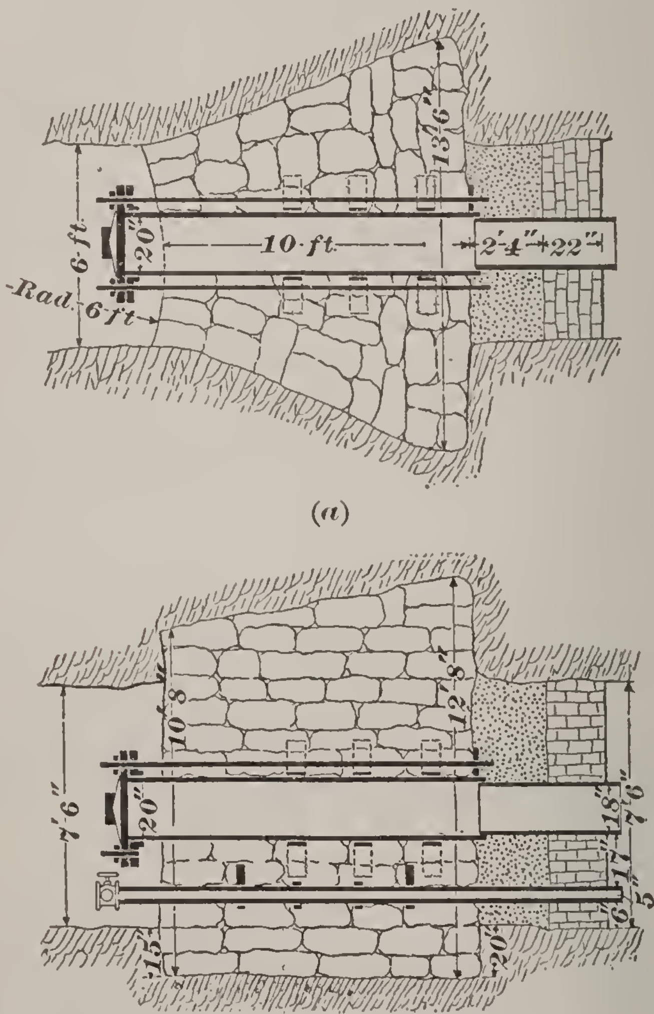


FIG. 2

small amount, but this soon practically ceased, showing that the manure had closed the leaks. A gauge in the head of the manway on this dam showed a pressure of 211 lb., which corresponded to a static head of 640 ft. of water. The total

pressure against the dam was something over 800 T., which it successfully resisted.

Reinforced concrete is largely used for mine dams, as these dams are strong, impermeable, quickly made, and of reasonable cost. Old rails, strap iron, etc. may be used for the reinforcing material.

In mine work, dams are used for retaining water in reservoirs, for diverting streams in placer mining, and for storing débris coming from placer mines in cañons or ravines.

Foundations for dams must be solid to prevent settling, and water-tight to prevent leakage under the base of the dam; whenever possible, the foundation should be solid rock. Gravel is better than earth, but when gravel is used sheet piling must be driven under the upper toe of the dam, to prevent water from seeping through the formation under the dam. Vegetable soil should be avoided, and all porous material, such as sand, gravel, etc. should be stripped off until hard pan or solid rock is reached.

Abutments are timber, masonry, or dry stonework structures at the ends of a dam. If possible, they should have a curved outline, and should be so placed that there is no possibility of the water overflowing them, or getting behind them during floods.

If the regular discharge from a dam takes place from the main face, the *discharge gates* may be arranged in connection with one of the abutments, or by means of a tunnel and culvert through the dam. In either case, some structure should be constructed above the outlet so as to prevent driftwood, brush, and other material from stopping the gates. When the discharge gates are placed at one side of the dam, they are usually arranged outside of the regular abutment, between it and another special abutment, the discharge being through a series of gates into a flume, ditch, or pipe.

Spillways or *waste ways*, are openings provided in a dam for the discharge of water during floods or freshets, or for the discharge of a portion not being used at any time. The spillway may be over the crest of the dam, or, where the topography favors such a construction, the main dam may be of sufficient height to prevent water from ever passing its crest and the

spillway arranged at another outlet over a lower dam. Waste ways, proper, are openings through the dam, to provide for the discharge of the large quantities of water that come down during freshets or floods.

Wooden Dams.—Wooden dams are constructed of round, sawed, or hewn logs. The timbers are usually at least 1 ft. square, or, if round, from 18 to 24 in. in diameter. A series of cribs from 8 to 10 ft. square are constructed by building up the logs log-house fashion and securing them together with treenails. The individual cribs are secured to one another with treenails or by means of bolts. The cribs are usually filled with loose rock to keep them in place, and in many cases are secured to the foundation by means of bolts. The dam is made water-tight by a layer of planking on the upper face and if the spillway is over the crest of the dam it will be necessary to plank the top of the cribs, and, in most cases, to provide an apron for the water to fall on. The apron may be set on small cribs, or on timbers projecting from the cribs of the dam itself.

Stone Dams.—Where cement or lime is expensive, and suitable rubble stone can be obtained, dams are frequently constructed without the use of mortar. The upper and lower faces of the dam should be of hammer-dressed stone, carefully bonded; sometimes the stones in the lower face of the dam are anchored by means of bolts. The dam can be made water-tight by means of a skin of planking on the upper face. In case water should pass over the crest of such a dam, much of it would settle through the openings in the stone into the interior of the dam, and thus subject the stones in the lower portion of the face to a hydrostatic pressure; for this reason, culverts or openings should be made through the lower portion of the dam, to discharge any such water. When such dams as this are constructed, the regular spillway is not placed over the face of the dam, but at some other point, and usually over a timber dam.

Earth Dams.—Earth dams are used for reservoirs of moderate height. They should be at least 10 ft. wide on top, a height of more than 60 ft. being unusual. When the material of which the dam is composed is not water-tight, it is sometimes

necessary to construct, in the center of the regular dam, a narrow dam of clay mixed with a certain proportion of sand. This puddle wall should not be less than from 6 to 8 ft. thick at the top and should have a slight batter on each side. It is constructed during the building of the dam, and should be protected from contact with the water by a considerable thickness of earth on the upper face. The upper face of an earthen dam is frequently protected by means of plank or a pavement of stone. The lower face is frequently protected by means of sod, or sod and willow trees. Sometimes earth dams are provided with a masonry core in place of the puddle wall, to render them water-tight.

Masonry Dams.—High masonry dams should always be designed by a competent hydraulic engineer. Masonry dams are not, as a rule, used for hydraulic mining, as the length of time during which the dam is required rarely warrants the expense of the construction of a masonry dam.

Débris Dams.—Dams or obstructions are sometimes placed across the bed of the stream to hold back culm, etc., from mines, and to prevent damage to the valleys below. They are made of stone, timber, or brush. No attempt is made to render these débris dams water-tight, as their only object is to retard the flow of the stream and to give it greater breadth of discharge, so that the water naturally drops and deposits the sediment that it is carrying. The sediment soon silts or fills up against the face of the dam, the area above the dam becoming a flat expanse or plain over which the water finds its way to the dam.

RESERVOIRS

In selecting a site for a reservoir, the points to be observed are:

A proper elevation above the point at which the water is required.

The total supply available, including observations as to the rainfall and snow fall.

The formation and character of the ground, with reference to the amount of absorption and evaporation.

PUMP MACHINERY

Pumps are used for unwatering mines, handling water at placer mines, irrigation, water-supply systems, boiler feeds, etc. For unwatering mines, two general systems of pumping are used: In one, the pump is placed in the mine and is operated by a motor on the surface, the power being transmitted through a line of moving rods; in the other, both the motor and the pump are placed in the mine, the motor being an engine driven by steam, compressed air, hydraulic motor, or an electric motor.

Cornish Pumps.—Any method of operating pumps by rods is commonly called a *Cornish system.* This system requires no steam line down the shaft and is independent of the depth of water in the mine, so that the pump is not stopped by the drowning of a mine, but the moving rods are a great inconvenience in the shaft, and they absorb much of the power through friction.

Simple and Duplex Pumps.—In the *simple pump*, a steam cylinder is connected directly to a water cylinder, and the steam valves are operated by tappets. Such a pump is more or less dependent on inertia at certain points of the stroke to insure the motion of the valves, hence will not start from any place and is liable to become stalled at times. In the *duplex pump*, two steam cylinders and two water cylinders are arranged side by side, and the valves are so placed that when one piston is at mid-stroke it throws the steam valve for the other cylinder, etc. With this arrangement, the pump will start from any point, and can never be stalled for lack of steam, due to the position of the valves. Ordinarily, duplex pumps are to be preferred for mine work.

The *packing* for the water piston of a pump may be either inside or outside. As a rule, inside-packed pumps should be avoided in mines, because acid or gritty waters are liable to cut the packing and make the pumps leak in a very short time. For dipping work in single stopes or entries, small single or duplex outside-packed pumps may be used. It is generally best to operate such pumps by compressed air, for the exhaust will then be beneficial to the mine air. If steam is used, it is

frequently necessary to introduce a trap and remove entrained water from the steam before it enters the pump, and dispose of the exhaust by piping it out or condensing it. Such isolated steam pumps are about the most wasteful form of steam-driven motor in existence.

For sinking, center-packed single or duplex pumps are usually employed, the duplex style being the better. For station work, where much water is to be handled, large compound, or triple-expansion, condensing, duplex pumping engines are employed. They may, or may not, be provided with cranks and a flywheel; engineers differ greatly upon this point, but as a rule, for very high lifts and great pressures, the flywheel is used.

Capacity of Pumps and Horsepower Required to Raise Water.—To find the capacity of pumps and the horsepower required to raise water any distance,

Let Q = cubic feet of water per minute;
G = U. S. gallons per minute;
G' = U. S. gallons per hour;
d = diameter of cylinder, in inches;
l = stroke of piston, in inches;
N = number of single strokes per minute;
v = speed of piston, in feet per minute;
W = weight moved, in pounds per minute;
P = pressure, in pounds per square feet $= 62.5 \times H$;
p = pressure, in pounds per square inch $= .433 \times H$;
H = height of lift, in feet;
H. P. = horsepower.

Then,
$$Q = \frac{\pi}{4} \times \frac{d^2}{144} \times \frac{lN}{12} = .0004545Nd^2l$$

$$G = \frac{\pi}{4} \times \frac{Nd^2l}{231} = .0034Nd^2l. \quad G' = .204Nd^2l$$

The diameter of piston required for a given capacity per minute will be

$$d = 46.9\sqrt{\frac{Q}{Nl}} = 17.15\sqrt{\frac{G}{Nl}}, \text{ or } d = 13.54\sqrt{\frac{Q}{v}} = 4.95\sqrt{\frac{G}{v}}$$

The actual capacity of a pump will vary from 60% to 95% of the theoretical capacity, depending on the tightness of the piston, valves, suction pipe, etc.

$$\text{H. P.}=\frac{QP}{33{,}000}=\frac{QH\times 144\times .433}{33{,}000}=\frac{QH}{529.2}=\frac{Gp}{1{,}714.5}$$

The actual horsepower required will be considerably greater than the theoretical, on account of the friction in the pump; hence, at least 20% should be added to the power for friction and usually about 50% more is added to cover leaks, etc., so that the actual horsepower required by the pump is about 70% more than the theoretical.

Limit of Suction.—Theoretically, a perfect pump will raise water to a height of nearly 34 ft. at the sea level; but owing to the fact that a perfect vacuum can never be attained with the pump, that the water always contains more or less air, and that more or less watery vapor will form below the piston, it is never possible to reach this theoretical limit, and, in practice, it is not possible to draw water much, if any, over 30 ft. at the sea level, even when the water is cold. Warm water cannot be lifted as high as cold water, because a larger amount of watery vapor forms. With boiler feed-pumps handling hot water, the water should flow to the pumps by gravity.

Power Pumps.—Where comparatively small amounts of water are to be handled and power is available, belt-driven power pumps are very much more efficient than small steam pumps. Where water is to be delivered from isolated workings to the sumps for the large station pumps, electrically driven power pumps are far more efficient than steam pumps.

Miscellaneous Forms of Water Elevators.—In the *jet pump* the energy of the jet of water is utilized for raising a larger volume through a small distance, or a mixture of water and solid material through a short distance.

The *pulsometer* consists of two chambers in a large casting, with suitable automatic valves arranged at the top and bottom of the chambers. Steam is introduced into one of the chambers, then the valve at the top closed. As this steam condenses, it forms a vacuum that draws water from the suction into the chamber. When the chamber is filled with water, steam is again introduced and forces the water out through the discharge pipe. The operation is then repeated, more water being drawn in by the condensation of the steam.

Air-lift pumps have not been successful as mine pumps, owing to the ratio between the part immersed and the lift.

In *centrifugal pumps*, the height of lift depends on the tangential velocity of the revolving disk of pump and the quantity of water discharged, and is proportional to the area of the discharge orifices at the circumference of the disk. The most efficient total lift for the centrifugal pump is, approximately, 17 ft., and for small lifts the centrifugal pump is much more efficient than any style of piston pump. For a given lift, the total efficiency of a centrifugal pump increases with the size of the pump. Centrifugal pumps are always designated by the size of their outlet, as, for instance, a 2-in. or 4-in. pump, meaning a 2-in. or 4-in. discharge pipe. Under the most favorable circumstances, the efficiency of the centrifugal pump may be practically 70%; that is, the pump may do an amount of work upon the water that is theoretically equal to 70% of the power furnished to the pump. Pumping engines working against high heads, and operated by the most improved class of engines, may attain an efficiency of practically 85%.

Where only a limited amount of water collects in the mine workings, it is frequently removed by means of a special *water bucket* or *water car* during the hours that the hoisting engine would otherwise be idle. Where very large amounts of water are to be removed, it has also been found economical to remove them by means of special water buckets; especially in the case of deep shafts.

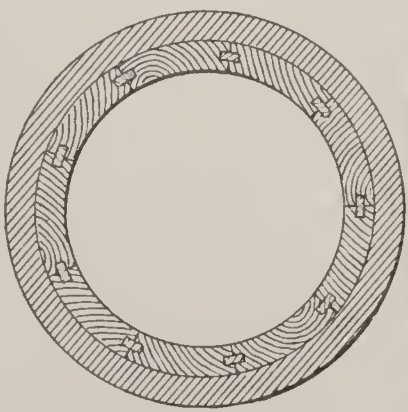


FIG. 1

FIG. 2

Pumps for Acid Waters.—Where mine waters are acid in their nature, brass or brass-lined pumps are usually employed. The pipes for such pumps should be of brass or copper tubing, or should be lined with some substance that will not be affected by the acid of the water. Sometimes wooden linings are

employed, placed as shown in Fig. 1, which is a section of the pipe with the lining complete. In Fig. 2 is shown a cross-section of one of the individual boards used in the lining. These boards are usually made of pine about $\frac{3}{8}$ in. thick, and are grooved on each end as shown. They are sprung in so as to complete a circle on the inside of the pipe, and then long, thin, wooden keys are driven into the grooves. When the water is allowed to go into the pipes, the linings swell and make all joints perfectly tight. Elbows and other crooked sections are lined with sheet lead beaten in with a mallet.

STEAM

FUELS

Classification of Coals.—Coals may be broadly divided into two classes: anthracite, or hard, coal, and bituminous, or soft, coal. The subdivisions given, however, are entirely arbitrary, as the different varieties of coal are found to shade insensibly into one another.

Anthracite, or *hard coal*, which has a specific gravity of 1.30 to 1.70, is the densest, hardest, and most lustrous of all varieties. It burns with little flame and no smoke, but gives a great heat, and contains very little volatile combustible matter. Its color is deep black and shining; sometimes it is iridescent; its fracture is conchoidal. *Semianthracite coal* is not so dense nor so hard as the true anthracite; its percentage of volatile combustible matter is somewhat greater, and it ignites more readily.

Bituminous, or *soft*, *coal*, which has a specific gravity of 1.25 to 1.40, is generally brittle. It has a bright pitchy or greasy luster, and is rather fragile as compared with anthracite. It burns with a yellow smoky flame, and gives, on distillation, hydrocarbon oils or tar. Under the term bituminous are included a number of varieties of coal that differ materially under the action of heat, giving rise to the general classification Coking or caking coals, and free-burning coals. *Semibituminous*

coal has the same general characteristics as the bituminous, although it is usually not so hard, and its fracture is more cuboid. The percentage of volatile combustible matter is less. It kindles readily, burns quickly with a steady fire, and is much valued as a steam coal.

Coking coals are those that become pasty or semiviscid in the fire; and, when heated in a closed vessel, become partially fused and agglomerate into a mass of coherent coke. This property of coking may, however, become greatly impaired, if, indeed, not entirely destroyed, by weathering. *Free-burning coals* have the same general characteristics as the coking coals, but they burn freely without softening, and do not fuse or cake together in any sensible degree.

Splint coal has a dull black color, and is much harder and less frangible than the coking coal. It is readily fissile, like slate, but breaks with difficulty on cross-fracture. It ignites less readily, but makes a hot fire, constituting a good house coal.

Cannel coal differs from the ordinary bituminous coal in its texture. It is compact, with little or no luster and without any appearance of a banded structure. It breaks with a smooth conchoidal fracture, kindles readily, and burns with a dense smoky flame. It is rich in volatile matter, and makes an excellent gas coal. Its color is dull black and grayish-black.

Lignite, or *brown coal*, often has a lamellar or woody structure; is sometimes pitch black, but more often rather dull and brownish black. It kindles readily and burns rather freely with a yellow flame and comparatively little smoke, but it gives only a moderate heat. It is generally non-coking. The percentage of moisture present is invariably high—from 10% to 30%.

Composition of Coals.—A proximate analysis determines the proportion of those products of a coal having the most important bearing on its uses. These substances as usually presented are: moisture, or water, volatile combustible matter, fixed carbon, sulphur, and ash. In addition to these, the following physical properties are generally given: color of ash, specific gravity and strength or hardness. The determination of these eight factors gives a fair general idea of the adaptabilities of a coal.

Moisture, or water, in coal, has no fuel value.

Volatile combustible matter is an important constituent of coal, the amount and quality deciding whether a coal is suitable for the manufacture of illuminating gas. The coking of coal also is largely dependent on this constituent.

The *fixed carbon* is the principal combustible constituent in coal, and, in bituminous and semibituminous coals, the steaming value is in proportion to the percentage of fixed carbon.

Sulphur will burn and develop heat and is not inert like moisture and ash; but it corrodes grates and boilers. In the blast furnace, it injures iron, and produces a hot short pig, and is objectionable in coal for forge use. For gas making, the sulphur must be removed.

Ash is an inert constituent, which means that 20 lb. of weight must be handled and 20 lb. loss per T. of coal for each per cent. of ash present. The color of the ash furnishes a rough estimate of the amount of iron contained in a fuel. Iron in an ash makes it more fusible, and increases its tendency to clinker.

The *specific gravity* is an important factor when there is restriction of space, as on railway cars and in ship bunkers. A given bulk of anthracite coal will weigh from 10% to 15% more than the same bulk of bituminous coal, so that from 10% to 15% more pounds of fuel can be carried in the same place.

Strength or *hardness* is valuable in preventing waste. A very soft coal is shipped in lump. Strength is a requisite for the use of raw coal in the blast furnace, and also to prevent excessive loss of coal through the grates in ordinary furnaces.

Coke is the fixed carbon of a coal, a fused and porous product produced by the distillation of the gaseous constituent. For metallurgical use, it should be firm, tough, and bright, with a sonorous ring, and should contain not over 1% of sulphur. For blast-furnace use, a dense coke is objectionable, and the best is the one with the largest cell structure and the hardest cell wall. A high percentage of volatile hydrocarbon is, as a rule, necessary for a good coking coal.

The essentials of a good *gas coal* are a low percentage of ash, say 5%, and of sulphur, say $\frac{1}{2}$ of 1%, a generous share, say 37% to 40% of volatile matter, charged with rich illuminating hydrocarbons. It should yield, under present retort practice,

85 candle-feet to the pound carbonized. It should be sufficiently dense to bear transportation well, so that, when carried long distances, it will not arrive at its destination largely reduced to slack or fine coal of the consistency of sand; and it should possess coking qualities that will bring from the retorts, after carbonization, about 60% of clean, strong, bright coke.

A good coal for *blacksmith purposes* should have a high heating power, should contain a very small amount of sulphur, if any, should coke sufficiently to form an arch on the forge, and should also be low in ash.

The analysis of a coal does not necessarily determine its value or the uses to which it can be put. However, for examining the analyses given in the accompanying tables, certain standards may be adopted as showing in a general way about what the analysis of coal should be for certain purposes. For steam purposes, the semibituminous coals have established reputations; for gas coals, that from Youghiogheny, Pa., is well known; for blacksmiths, Broad Top and Tioga County, Pa., coals are standards; while for coking, Connellsville is recognized as the best.

Heating Formulas.—A *British thermal unit* (B. T. U.) is the quantity of heat required to raise the temperature of 1 lb. of water 1° F. at or near the temperature of maximum density, 39.1° F.

A *calorie* (cal.) is the quantity of heat required to raise the temperature of 1 Kg. of water 1° C. at or about 4° C.

A *pound calorie* is the quantity of heat necessary to raise the temperature of 1 lb. of water 1° C.

$$\begin{aligned} 1 \text{ French cal.} &= 3.968 \text{ B. T. U.} \\ 1 \text{ B. T. U.} &= .252 \text{ cal.} \\ 1 \text{ lb. cal.} &= \tfrac{9}{5} \text{ B. T. U.} = .4536 \text{ cal.} \end{aligned}$$

The heating value of any coal may be calculated from its ultimate analysis, with a probable error not exceeding 2%, by Dulong's formula:

$$\text{Heating value per pound} = 146\,C + 620\left(H - \frac{O}{8}\right),$$

in which C, H, and O are, respectively, the percentages of carbon, hydrogen, and oxygen.

PROXIMATE ANALYSES AND HEATING VALUES OF AMERICAN COALS

Coal	Moisture Per Cent.	Volatile Matter Per Cent.	Fixed Carbon Per Cent.	Ash Per Cent.	Sulphur Per Cent.	Heating Value per Pound Coal B. T. U.	Volatile Matter Per Cent. of Combustible	Fixed Carbon Per Cent. of Combustible	Heating Value per Pound Combustible B. T. U.	Theoretical Evaporation From and at 212° per Pound Combustible
Anthracite										
Northern Coal Field	3.42	4.38	83.27	8.20	.73	13,160	5.00	95.00	14,900	15.42
East Middle Coal Field	3.71	3.08	86.40	6.22	.58	13,420	3.44	96.56	14,900	15.42
West Middle Coal Field	3.16	3.72	81.59	10.65	.50	12,840	4.36	95.64	14,900	15.42
Southern Coal Field	3.09	4.28	83.81	8.18	.64	13,220	4.85	95.15	14,900	15.42
Semianthracite										
Loyalsock Field	1.30	8.10	83.34	6.23	1.63	13,920	8.86	91.14	15,500	16.05
Bernice Basin	.65	9.40	83.69	5.34	.91	13,700	10.98	89.02	15,500	16.05
Semibituminous										
Broad Top, Pa.	.79	15.61	77.30	5.40	.90	14,820	17.60	82.40	15,800	16.36
Clearfield Co., Pa.	.76	22.52	71.82	3.99	.91	14,950	24.60	75.40	15,700	16.25
Cambria Co., Pa.	.94	19.20	71.12	7.04	1.70	14,450	22.71	77.29	15,700	16.25
Somerset Co., Pa.	1.58	16.42	71.51	8.62	1.87	14,200	20.37	79.63	15,800	16.36
Cumberland, Md.	1.09	17.30	73.12	7.75	.74	14,400	19.79	80.21	15,800	16.36
Pocahontas, Va.	1.00	21.00	74.39	3.03	.58	15,070	22.50	77.50	15,700	16.25

New River, W. Va.	.85	17.88	77.64	3.36	.27	15,220	18.95	81.05	15,800	16.36
Bituminous										
Connellsville, Pa.	1.26	30.12	59.61	8.23	.78	14,050	34.03	65.97	15,300	15.84
Youghiogheny, Pa.	1.03	36.50	59.05	2.61	.81	14,450	38.73	61.27	15,000	15.53
Pittsburg, Pa.	1.37	35.90	52.21	8.02	1.80	13,410	41.61	58.39	14,800	15.32
Jefferson Co., Pa.	1.21	32.53	60.99	4.27	1.00	14,370	35.47	64.53	15,200	15.74
Middle Kittaning Seam, Pa.	1.81	35.33	53.70	7.18	1.98	13,200	40.27	59.73	14,500	15.01
Upper Freeport Seam, Pa. and O.	1.93	35.90	50.19	9.10	2.89	13,170	43.59	56.41	14,800	15.32
Thacker, W. Va.	1.38	35.04	56.03	6.27	1.28	14,040	39.33	60.67	15,200	15,74
Jackson Co., Ohio	3.83	32.07	57.60	6.50		13,090	35.76	64.24	14,600	15.11
Brier Hill, Ohio	4.80	34.60	56.30	4.30		13,010	38.20	61.80	14,300	14.80
Hocking Valley, Ohio	6.59	34.97	48.85	8.00	1.59	12,130	42.81	57.19	14,200	14.70
Vanderpool, Ky.	4.00	34.10	54.60	7.30		12,770	38.50	61.50	14,400	14.91
Muhlenberg Co., Ky.	4.33	33.65	55.50	4.95	1.57	13,060	38.86	61.14	14,400(?)	14.91
Scott Co., Tenn.	1.26	35.76	53.14	8.02	1.80	13,700	34.17	65.83	15,100(?)	15.63
Jefferson Co., Ala.	1.55	34.44	59.77	2.62	1.42	13,770	37.63	62.37	14,400(?)	14.91
Big Muddy, Ill.	7.50	30.70	53.80	8.00		12,420	36.30	63.70	14,700	15.22
Mt. Olive, Ill.	11.00	35.65	37.10	13.00		10,490	47.00	53.00	13,800	14.29
Streator, Ill.	12.00	33.30	40.70	14.00		10,580	45.00	55.00	14,300	14.80
Missouri.	6.44	37.57	47.94	8.05		12,230	43.94	56.06	14,300(?)	14.80
Lignite and Lignitic Coals										
Iowa.	8.45	37.09	35.60	18.86		8,720	51.03	48.97	12,000(?)	12.42
Wyoming.	8.19	38.72	41.83	11.26		10,390	48.07	51.93	12,900(?)	13.35
Utah.	9.29	41.97	44.37	3.20	1.18	11,030	48.60	51.40	12,600(?)	13.04
Oregon lignite.	15.25	42.98	33.32	7.11	1.66	8,540	54.95	45.05	11,000(?)	11.39

$$\text{Heat in pound calorie} = 8{,}080\ C + 34{,}462\left(H - \frac{O}{8}\right)$$

$$\text{or} = 8{,}080\ C + 34{,}462\left(H - \frac{O}{8}\right) + 2{,}250\ S$$

$$\text{Heat in B. T. U.} = 14{,}650\ C + 62{,}100\left(H - \frac{O}{8}\right)$$

in which C, O, H, and S represent the weights of carbon, oxygen, hydrogen, and sulphur in 1 lb. of the substance.

COMPOSITION OF FUELS

Description	Carbon Per Cent.	Hydrogen Per Cent.	Oxygen Per Cent.	Nitrogen Per Cent.	Sulphur Per Cent.	Ash Per Cent.
Anthracite						
France	90.9	1.47	1.53	1.00	.80	4.3
Wales	91.7	3.78	1.30	1.00	.72	1.5
Rhode Island	85.0	3.71	2.39	1.00	.90	7.0
Pennsylvania	78.6	2.50	1.70	.80	.40	14.8
Semibituminous						
Maryland	80.0	5.00	2.70	1.10	1.20	8.3
Wales	88.3	4.70	.60	1.40	1.80	3.2
Bituminous						
Pennsylvania	75.5	4.93	12.35	1.12	1.10	5.0
Indiana	69.7	5.10	19.17	1.23	1.30	3.5
Illinois	61.4	4.87	35.42	1.41	1.20	5.7
Virginia	57.0	4.96	26.44	1.70	1.50	8.4
Alabama	53.2	4.81	32.37	1.62	1.30	6.7
Kentucky	49.1	4.95	41.13	1.70	1.40	7.2
Cape Breton	67.2	4.26	20.16	1.07	1.21	6.1
Vancouver Island	66.9	5.32	8.76	1.02	2.20	15.8
Lancashire gas coal	80.1	5.50	8.10	2.10	1.50	2.7
Boghead cannel	63.1	8.90	7.00	.20	1.00	19.8
Lignite						
California brown	49.7	3.78	30.19	1.00	1.53	13.8
Australian brown	73.2	4.71	12.35	1.11	.63	8.0
Petroleum						
Pennsylvania (crude)	84.9	13.70	1.40			
Caucasian (light)	86.3	13.60	.10			
Caucasian (heavy)	86.6	12.30	1.10			
Refuse	87.1	11.70	1.20			

STEAM BOILERS

The steam boiler that will be the most suitable for a certain mine will depend on the nature of the feedwater, the cost of fuel, and the amount of steam required. When the acid water from the mine is used for feedwater and fuel is cheap, either the plain cylindrical or the flue boiler is used, because it is simple in construction and can therefore be easily cleaned and cheaply replaced when eaten by the mine water. The tubular or locomotive type is used where good water can be obtained; except in the best-equipped plants, where the water-tube boiler is used. Feedwater taken from the mine, or containing acid, should be neutralized by lime or soda before being used. In case it contains minerals in solution, a feedwater separator should be employed to precipitate the mineral substance before the water is allowed to enter the boiler.

The *heating surface* of a boiler is the portion of the surface exposed to the action of flames and hot gases. This includes, in the case of the multi-tubular boiler, the portions of the shell below the line of brickwork, the exposed heads of the shell, and the interior surface of the tubes. In the case of a water-tube boiler, the heating surface comprises the portion of the shell below the brickwork, the outer surface of the headers, and outer surface of tubes.

Horsepower of Boilers.—The *horsepower* of a boiler is a measure of its capacity for generating steam. Boilermakers usually rate the horsepower of their boilers as a certain fraction of the heating surface; but this is a very indefinite method, for with the same heating surface, different boilers of the same type may, under different circumstances, generate different quantities of steam.

In order to have an accurate standard of boiler power, the American Society of Mechanical Engineers has adopted as a standard horsepower an evaporation of 30 lb. of water per hour from a feedwater temperature of 100° F. into steam at 70 lb. gauge pressure, which is considered equivalent to 34.5 units of evaporation; that is, to 34.5 lb. of water evaporated from a feedwater temperature of 212° F. into steam at the same temperature.

High-Pressure Steam.—A calculation of the power that coal possesses, compared with the useful work which steam engines exert, shows that probably in the very best engines not one-tenth of the power is converted into useful work, and in some very bad engines, probably not one one-hundredth. Whatever pressure may be available at the steam boiler, a certain amount is absorbed in overcoming the resistance of the engine and without doing any useful work. Then, again, the amount of work that it is possible to get out of a given quantity of steam depends on the difference between the temperature at the commencement of the stroke and the temperature at the end of the stroke.

There is a limit as to how low the temperature can be at the end, therefore, as the commencing temperature is raised the available difference is increased. The advantages of high-pressure steam may be shown by taking a fixed temperature in the condenser of say, 100° F., then initial temperatures when the steam enters the cylinder, the temperature of varying amounts, and the theoretic efficiency of that steam can be determined. At atmospheric pressure, there is an efficiency of 16.6%.

EFFICIENCY OF STEAM AT VARIOUS PRESSURES

Steam Pressure Pounds	Efficiency Per Cent.	Steam Pressure Pounds	Efficiency Per Cent.
10	20.0	100	29.8
20	22.1	125	31.1
30	23.7	150	32.2
40	25.0	200	33.9
50	26.1	250	35.3
60	27.0	300	36.5
80	28.6		

In practice, only a certain proportion of the theoretic power of steam can be obtained, and that proportion varies with the pressure of the steam. The advantages of high-pressure steam are not yet sufficiently appreciated. It is not merely the difference between 60 lb. and 120 lb. Suppose we use steam

at 60 lb.; probably we shall get 50 lb. at engine, and resistances of engine will absorb 10 lb., leaving 40 lb. Now, suppose we use 120 lb., we can get at engine 110 lb., and if resistances of engine absorb 10 lb., we shall have 100 lb. as against 40 lb.

By *expansion of steam* is meant that at a certain point of the stroke, the steam supply from the boiler to the cylinder is shut off and the steam already within the cylinder performs the remainder of the stroke unaided.

Incrustation.—Nearly all waters contain foreign substances in a greater or less degree, and though this may be a small amount in each gallon, it becomes of importance where large quantities are evaporated. For instance, a 100-H. P. boiler evaporates 30,000 lb. of water in 10 hr. or 390 T. per mo.; in comparatively pure water there should be 88 lb. of solid matter in that quantity, and in many kinds of spring water as much as 2,000 lb.

The nature and hardness of the scale formed will depend on the kind of substances held in solution and suspension. Analyses of incrustations show that carbonate and sulphate of lime form the larger part of all ordinary scale, that from carbonate being soft and granular, and that from sulphate, hard and crystalline. Organic substances in connection with carbonate of lime will also make a hard and troublesome scale. The causes of incrustation are:

1. Deposition of suspended matter.
2. Deposition of salts from concentration.
3. Deposition of carbonates of lime and magnesia, by boiling off carbonic acid, which holds them in solution.
4. Deposition of sulphates of lime, because sulphate of lime is soluble in cold water, less soluble in hot water, insoluble above 270° F.
5. Deposit of magnesia, because magnesium salts decompose at high temperatures.
6. Deposition of lime soap, iron soap, etc. formed by saponification of grease.

Incrustation may be prevented by the following methods:

1. Filtration.
2. Blowing off.

3. Use of internal collecting apparatus, or devices, for directing the circulation.
4. Heating feedwater.
5. Chemical or other treatment of water in boiler.
6. Introduction of zinc in boiler.
7. Chemical treatment of water outside of boiler.

INCRUSTATION REMEDIES

Troublesome Substances	Trouble	Remedy or Palliation
Sediment, mud, clay, etc.	Incrustation	Filtration; blowing off
Readily soluble salts....	Incrustation	Blowing off
Bicarbonate of lime, magnesia, and iron....	Incrustation	Heating feed; addition of caustic soda, lime, or magnesia, etc.
Sulphate of lime........	Incrustation	Addition of carbonate of soda, barium chloride, etc.
Chloride and sulphate of magnesium...........	Corrosion	Addition of carbonate soda, etc.
Carbonate of soda in large amounts........	Priming	Addition of barium chloride, etc.
Acid in mine water......	Corrosion	Alkali
Dissolved carbonic acid and oxygen..........	Corrosion	Heating feed; addition of caustic soda, slaked lime, etc.
Grease, from condensed water...............	Corrosion	Slaked lime and filtering; substitute mineral oil
Organic matter, sewage.	Priming	Precipitate with alum or ferric chloride, and filter
Organic matter.........	Corrosion	Precipitate with alum or ferric chloride, and filter

Prevention of Incrustation.—The incrustation of boilers may be prevented by adding various substances to the feedwater. Oak, hemlock, sumac, catechu, logwood, and other barks and

woods, are effective in waters containing carbonates of lime or magnesia, by reason of their tannic acid, but are injurious to the iron and not to be recommended.

Molasses, cane juice, vinegar, fruits, distillery slops, etc., have been used, but the acetic acid that they contain is even more injurious to the iron than tannic acid, while the organic matter forms a scale with sulphate of lime when it is present.

Milk of lime and metallic zinc have been used with success in waters charged with bicarbonate of lime, reducing the bicarbonate to the insoluble carbonate.

Barium chloride and milk of lime are said to be used, with good effect at Krupp's works, in Prussia, for waters impregnated with gypsum.

Soda ash and other alkalies are very useful in waters containing sulphate of lime, by converting it into a carbonate, and so forming a soft scale that is easily cleaned. But when used in excess they cause foaming, particularly where there is oil coming from the engine, with which they form soap. All soapy substances are objectionable for the same reason.

Petroleum has been much used of late years; it acts best in waters in which sulphate of lime predominates. Sulphate of lime is the injurious substance in nearly all mine waters, and petroleum, when properly prepared, is a good preventive of scale and pitting. Crude petroleum should not be used, as it sometimes helps to form a very injurious scale. Refined petroleum, on the other hand, is useless, as it vaporizes at a temperature below that of boiling water. Therefore, only such preparations should be used as will not vaporize below 500° F.

Tannate of soda is a good preparation for general use, but in waters containing much sulphate, it should be supplemented by a portion of carbonate of soda or soda ash.

A decoction from the leaves of the eucalyptus is found to work well in some waters in California.

For muddy water, particularly if it contain salts of lime, no preventive of incrustation will prevail except filtration, and in almost every instance the use of a filter, either alone or in connection with some means of precipitating the solid matter from solution, will be found very desirable.

In all cases where impure or hard waters are used, frequent blowing from the mud-drum is necessary to carry off the accumulated matter, which if allowed to remain would form scale.

When boilers are coated with a hard scale, diffcult to remove, the addition of $\frac{1}{4}$ lb. caustic soda per horsepower, and steaming for some hours, according to the thickness of the scale, will greatly facilitate the cleaning, rendering the scale soft and loose. This should be done, if possible, when the boilers are not otherwise in use.

STEAM ENGINES

Requirements of a Good Steam Engine.—A good steam engine should be as direct acting as possible; that is, the connecting parts between the piston and the crank-shaft should be few in number, as each part wastes some power. Formerly, beam engines were in general use and were suitable for pumping when the pump was at one end of the beam and the piston at the other. Few of modern colliery engines, however, are thus equipped. The moving parts of an engine should be strong, to resist strains, and light, so as to offer no undue resistance to motion; parts moving upon each other should be well finished, to reduce resistances to a minimum; the steam should get into the cylinder easily at the proper time, and the exhaust should leave the cylinder as exactly and as easily. The steam pipes supplying steam should have an area one-tenth the combined areas of the cylinders they supply, and exhaust pipes should be somewhat larger. The cylinder, steam pipes and boiler should be well protected. The engine should be capable of being started and stopped and reversed easily and quickly.

Rule.—*To find the indicated horsepower developed by an engine, multiply the mean effective pressure per square inch, the area of the piston, the length of stroke, and the number of strokes per minute; this gives the work per minute in foot-pounds. Divide the product by 33,000.*

Let I. H. P. = indicated horsepower of engine;
P = M. E. P., in pounds per square inch;
A = area of piston, in square inches;
L = length of stroke, in feet;
N = number of strokes per minute.

Then, $$\text{I. H. P.} = \frac{PLAN}{33{,}000}$$

The number of strokes per minute is twice the number of revolutions per minute. For cxample, if an engine runs at a speed of 210 rev. per min., it makes 420 strokes per minute. A few types of engines, however, are single acting; that is, the steam acts on only one side of the piston, then the number of strokes per minute equals the number of revolutions per minute.

Approximate Determination of M. E. P.—To approximately determine the mean effective pressure, M. E. P., of an engine, when the point of apparent cut-off is known and the boiler pressure, or the pressure per square inch in the boiler from which the supply of steam is obtained, is given:

Rule.—*To find the M. E. P. of good, simple, non-condensing engines, add 14.7 to the gauge pressure, and multiply the result by the number opposite the fraction indicating the point of cut-off in the accompanying table. Subtract 17 from the product, and multiply by .9.*

Let p = gauge pressure;

k = a constant;

M. E. P. = mean effective pressure.

Then, $$\text{M. E. P.} = .9 \times [k(p + 14.7) - 17]$$

TABLE OF CONSTANTS

Cut-Off	Constant	Cut-Off	Constant	Cut-Off	Constant
$\frac{1}{6}$	.566	$\frac{3}{8}$	.771	$\frac{2}{3}$	.917
$\frac{1}{5}$	.603	$\frac{2}{5}$	.789	$\frac{7}{10}$	.926
$\frac{1}{4}$	.659	$\frac{1}{2}$	.847	$\frac{3}{4}$	.937
$\frac{3}{10}$	.708	$\frac{3}{5}$	.895	$\frac{4}{5}$	.944
$\frac{1}{3}$	.743	$\frac{5}{8}$	.904	$\frac{7}{8}$	.951

If the engine is a simple condensing one, subtract the pressure in the condenser instead of 17. The fraction indicating the point of cut-off is obtained by dividing the distance that the piston has traveled when the steam is cut off by the whole length of the stroke. For a $\frac{2}{3}$ cut-off, and 92 lb. gauge pressure in the boiler, the M. E. P. is, by the formula just given, $.9 \times [.917 \times (92 + 14.7) - 17] = 72.7$ lb. per sq. in.

COMPRESSED AIR

An *air compressor* consists essentially of a cylinder in which atmospheric air is compressed by a piston, the driving power being steam or water. Steam-driven compressors in ordinary use may be classed as follows:

(1) *Straight-line* type, in which a single horizontal air cylinder is set tandem with its steam cylinder, and provided with two flywheels; this pattern is generally adapted for compressors of small size.

(2) *Duplex* type, in which there are two steam cylinders, each driving an air cylinder, and coupled at 90° to a crank-shaft carrying a flywheel.

(3.) *Horizontal, cross-compound* engines, each steam cylinder set tandem with an air cylinder.

(4.) *Vertical, simple* or *compound* engines, with the air cylinders set above the steam cylinders.

(5.) *Compound* or *stage compressors*, in which the air cylinders themselves are compounded. The compression is carried to a certain point in one cylinder and successively raised and finally completed to the desired pressure in the others. They may be either of the straight-line or duplex form, with simple or compound steam cylinders.

The first three and the last classes are those commonly used for mine service. The principle of compound, or two-stage, air compression is recognized as applicable for even the moderate pressures required in mining, and the compressors of class 5 are frequently employed.

Transmission of Air in Pipes.—The actual discharge capacity of piping is not proportional to the cross-sectional area alone, that is, to the square of the diameter. Although the periphery is directly proportional to the diameter, the interior surface resistance is much greater in a small pipe than in a large one, because, as the pipe becomes smaller, the ratio of perimeter to area increases. Among the formulas in common use, perhaps the most satisfactory is that of D'Arcy. As adopted for compressed-air transmission, it takes the form:

$$D = c\sqrt{\frac{d^5(p_1 - p_2)}{w_1 l}},$$

in which D = volume of compressed air, in cubic feet per minute, discharged at final pressure;

c = coefficient varying with diameter of pipe, as determined by experiment;

d = diameter of pipe, in inches (the actual diameters of $1\frac{1}{4}$ in. and $1\frac{1}{2}$ in. pipe are 1.38 in. and 1.61 in., respectively; the nominal diameters of all other sizes may be taken for calculations);

l = length of pipe, in feet;

p_1 = initial gauge pressure, in pounds per square inch;

p_2 = final gauge pressure, in pounds per square inch;

w_1 = density of air, or its weight, in pounds per cubic foot, at initial pressure p_1.

The values of the coefficients c for piping up to 12 in. in diameter are given in the accompanying table. Some apparent discrepancies exist for sizes larger than 9 in. but they cause no very material differences in the result.

PIPING COEFFICIENTS

Size of Pipe Inches	Coefficient c	Size of Pipe Inches	Coefficient c	Size of Pipe Inches	Coefficient c
1	45.3	5	59.0	9	61.0
2	52.6	6	59.8	10	61.2
3	56.5	7	60.3	11	61.8
4	58.0	8	60.7	12	62.0

Loss of Pressure in Transmission.—In the accompanying table is given the loss of pressure in the transmission of compressed air, calculated for pipes 1,000 ft. long. For other lengths the loss varies directly as the length. The resistance is not varied by the pressure, only so far as changes in pressure vary the velocity. It increases about as the square of the velocity, and directly as the length. Elbows, short turns, and leaks in pipes tend to reduce the pressure in addition to the losses given in the table.

LOSS OF PRESSURE BY FLOW OF AIR IN PIPES

Velocity at Entrance		1-In. Pipe			2-In. Pipe			2½-In. Pipe			3-In. Pipe		
Meters per Second	Feet per Second	Loss of Pressure Pounds	Air per Min. Compressed to 60 Lb. Cubic Feet	Air per Min. Compressed to 80 Lb. Cubic Feet	Loss of Pressure Pounds	Air per Min. Compressed to 60 Lb. Cubic Feet	Air per Min. Compressed to 80 Lb. Cubic Feet	Loss of Pressure Pounds	Air per Min. Compressed to 60 Lb. Cubic Feet	Air per Min. Compressed to 80 Lb. Cubic Feet	Loss of Pressure Pounds	Air per Min. Compressed to 60 Lb. Cubic Feet	Air per Min. Compressed to 80 Lb. Cubic Feet
1	3.28	.1435	6	7	.0794	23	29	.0574	32	41	.0463	48	60
2	6.56	.6405	12	15	.3050	46	59	.2562	65	82	.2092	96	121
3	9.84	1.4545	18	22	.7216	69	88	.5818	97	124	.4880	144	182
4	13.12	2.5620	24	29	1.2566	93	117	1.0248	130	165	.8381	193	243
5	16.40	3.9345	29	37	1.9642	116	146	1.5738	163	207	1.3176	241	304
6	19.68	5.4225	35	44	2.7120	139	175	2.1690	195	247	1.8080	289	364
8	26.24	10.2480	47	59	5.0264	185	234	4.0992	260	330	3.3525	386	486
10	32.80	15.7380	59	74	7.8568	232	294	6.2952	326	413	5.2704	480	607
		4-In. Pipe			5-In. Pipe			6-In. Pipe			8-In. Pipe		
1	3.28	.0347	86	109	.0287	134	169	.0232	193	244	.0173	343	434
2	6.56	.1525	172	217	.1281	268	239	.1046	386	488	.0762	687	864
3	9.84	.3608	258	326	.2909	402	509	.2440	579	633	.1805	1,030	1,303
4	13.12	.6283	343	436	.5124	537	678	.4190	772	977	.3141	1,373	1,736
5	16.40	.9821	429	544	.7869	671	844	.6588	965	1,221	.4910	1,717	2,171
6	19.68	1.3560	515	653	1.0845	805	1,017	.9040	1,158	1,466	.6780	2,060	2,605
8	26.24	2.5132	687	871	2.0496	1,073	1,357	1.6762	1,544	1,954	1.2556	2,747	3,473
10	32.80	3.9284	859	1,088	3.1476	1,342	1,696	2.6352	1,931	2,443	1.9642	3,434	4,342

ELECTRICITY

ELECTRIC GENERATORS AND MOTORS

An *electric generator* is a machine for converting mechanical energy into electrical energy. An *electric motor* is a machine for converting electrical energy into mechanical energy. Either a generator or a motor may be called a *dynamo;* but this word is commonly used to denote generators only. Generators and motors may be divided into two general classes: those used with direct current and those used with alternating current.

Direct-Current Generators.—Direct-current generators are those that furnish a current always in the same direction. They have three essential parts: the *field magnet*, often called the *field*, the *armature*, and the *commutator;* the field is stationary but the armature revolves. The coils of wire on the field are called the *field coils*, and when properly connected make up the *field winding.* The individual coils on the armature are called the *armature coils*, and when properly connected make up the *armature winding.* The commutator consists of copper bars arranged in the form of a cylinder and revolves with the armature. Stationary *brushes*, usually of carbon, collect the current from the commutator; these brushes are held in *brush holders.*

Direct-Current Motors.—Direct-current motors are in general almost identical, so far as construction goes, with direct-current generators. Motors are often required to operate under very trying conditions, as for example, in mine haulage or pumping plants or on the ordinary street car. For this reason, their mechanical construction often differs from that of the generator so that the working parts will be enclosed as completely as possible, and thus protect them from dirt and injury. Practically all of the motors in use are operated from constant-pressure mains; i. e., the pressure at the terminals of the motor is practically constant, no matter what load it may be carrying. The various principal parts of a direct-current motor and a generator are similarly named.

None but very small motors or motors of special design should be started without some form of resistance in series with it to cut down the electric pressure of the line until the motor is up to speed. Such a resistance is called a *starting rheostat*, or *starting box*.

Motor Troubles.—If a *motor fails to start* when the controlling switch is closed, the trouble may be due to any one of several causes. There may be an open circuit, a short circuit, a wrong connection, the power may be off the line, or the trouble may be purely mechanical. If there is no power on the line, the lamps on the same line will be out. If the power is on the line, but there is no flash at the starting box when the handle is moved on and then off, the trouble is due to an open circuit. Open circuits that may be located by inspection are: defective switches, broken wires, loose or open connections, something under one of the brushes, brush stuck in the holder, brush dropped out of holder, or a blown fuse. With the exception of crosses in the wiring, short circuits are usually in the motor itself and require the attention of a skilled electrician.

Mechanical troubles that may interfere with the starting of a motor are sometimes overlooked. The more common are: too much load, bearings worn until the armature rubs the field magnet, sprung armature shaft, hot-box, tight belt or something in the gearing, lack of end play in the armature.

Excessive flashing at the brushes of a motor is called *sparking*. When a motor sparks badly the attention of an electrician should be called to it. The following are some of the common causes of sparking: Too much load, brushes improperly set, commutator rough or eccentric, dirty brushes or commutator, loose brushes, sprung armature shaft, low bearings, worn commutator, vibration, belt slipping. There are other causes, but they cannot be located readily except by an electrician.

Heating of a motor is usually due to overload. If a person cannot hold his hand firmly on any part of a motor, the machine is running too hot. Most modern motors will run continuously under an overload of 25% without serious heating and will run for 2 hr. under 50% overload without damage due to heating. These limits should not be exceeded.

Alternating-Current Generators.—An alternating-current generator, commonly called an *alternator*, is one that establishes a current which periodically reverses its direction. Alternators are now largely used for both lighting and power transmission, especially when the transmission is over long distances. Alternating current is specially suitable for long-distance work because it may be readily transformed from one pressure to another. In order to keep down the amount of copper in the line, a high line pressure must be used. Pressures much over 500 or 600 volts cannot be readily generated with direct-current machines, owing to the troubles likely to arise due to sparking at the commutator. An alternator requires no commutator; usually the armature is made stationary and the field revolving. Alternators are now built that generate as high as 8,000 or 10,000 volts directly. If a still higher pressure is required on the line, it can be easily obtained by the use of transformers.

Alternators may be divided into two classes: Single-phase and polyphase alternators. *Single-phase alternators* are so called because they set up a single alternating current. *Polyphase alternators* are so called because they deliver two or more alternating currents that differ in *phase;* that is, currents that do not reach their maximum nor their zero values of the same direction at the same instant.

Polyphase alternators are well adapted for power and lighting purposes in mines, especially for the operation of pumping and hoisting machinery, because the motors operated by them are simple in construction and not liable to get out of order.

Alternating-Current Motors.—Alternating-current motors may be divided into two classes: synchronous and induction motors. *Synchronous motors* are almost identical, so far as construction goes, with the corresponding alternator. *Induction motors* are so called because the current is induced in the armature instead of being led into it from some outside source. The stationary part of an induction motor is usually called the *stator* and the revolving part, the *rotor*.

Induction motors possess many advantages for mine work. One is the absence of the commutator or any kind of sliding contacts whatever. Such motors can therefore operate with

absolutely no sparking—a desirable feature for mine work. The motors are also very simple in construction, and are not liable to get out of order, but, like direct-current motors, they should not be overloaded

Transformers.—Transformers are used to change an alternating current from a higher to a lower pressure, or vice versa, with a corresponding change in current. Transformers used for raising the voltage are known as *step-up transformers;* those used for lowering the pressure are known as *step-down transformers.* The transformer consists of a laminated iron core upon which are wound two coils of wire that have no connection with each other. One of these coils, called the *primary*, is connected to the mains; the other coil, called the *secondary*, is connected to the circuit to which current is delivered. The core and coils are contained in an iron case usually filled with oil.

ELECTRIC CIRCUITS

The path through which a current flows is generally spoken of as an *electric circuit.* This path may be made up of a number of parts. For example, the line wires may constitute part of the circuit, and the remainder may be composed of lamps, motors, resistances, etc. In practice, the two kinds of circuits most commonly met with are those in which the different parts of the circuit are connected in series and those in which the different parts of the circuit are connected in multiple or parallel.

Series Circuits.—In the series circuits any two adjacent parts are connected in tandem, so that the current passing through one part also passes through the other parts. Fig. 1 (*a*) represents such a circuit in which the current passes from the generator D at the + side through the arc lamps a, through the incandescent lamps l, through the motor m and the resistance r, back to the generator D. The most common use of this system is in connection with arc lamps.

The objections to this system of distribution for general work are that the breaking of the circuit at any point cuts off the current from all parts of the circuit; also, the pressure

generated by the dynamo has to be very high if many pieces of apparatus are connected in series. In such a system, the dynamo is provided with an automatic regulator that increases or decreases the voltage of the machine, so that the current in the circuit is kept constant, no matter how many lamps or other devices are in operation. For this reason, such circuits are often spoken of as *constant-current circuits*.

A series circuit should never be opened at any point unless it is known that there is no current in the line. If it is desired

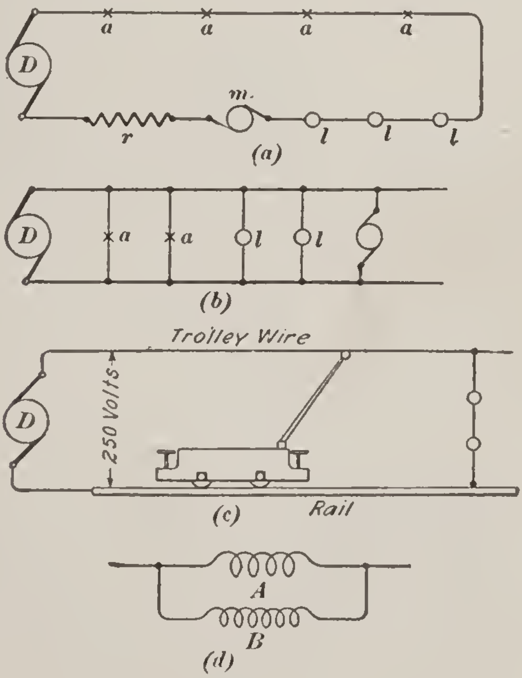


FIG. 1

to disconnect an arc lamp, for instance, from a series circuit, one end of a short wire, called a *jumper*, should be connected to the line wire on each side of the lamp, so that the current may pass through the jumper. Then the lamp may be disconnected.

Parallel Circuits.—In parallel circuits, the different pieces of apparatus are connected side by side, or in parallel, across the main wires, as shown in Fig. 1 (*b*). In this case, the generator *D* supplies current through the mains to the arc

lamps *a*, incandescent lamps *l*, and motor *m*. This system is more widely used as the breaking of the circuit through any one piece of apparatus will not stop the current through the other parts. Incandescent lamps are connected in this way almost entirely. Street cars and mining locomotives are operated in the same way, the trolley wire constituting one main and the track the other, as shown in Fig. 1 (*c*). By adopting this system, any car can move independently of the others, and the current may be turned off and on at will. In all these systems of parallel distribution, the pressure of the generator is maintained constant no matter what current the generator may be delivering. In mine-haulage plants, the pressure is usually 250 or 500 volts, the former being generally preferred as being less dangerous. Lamps may also be connected in series-multiple, as shown in Fig. 1 (*c*). Here the two 125-volt lamps *l* are connected in series across the 250-volt circuit. Such an arrangement is frequently used in mines when lamps are operated from the haulage circuit. Parallel circuits are called *constant-potential circuits*, to distinguish them from the *constant-current circuit* mentioned previously.

Shunt.—When one circuit *B*, Fig. 1 (*d*), is connected across another *A*, so as to form, as it were, a by-pass, or side-track,

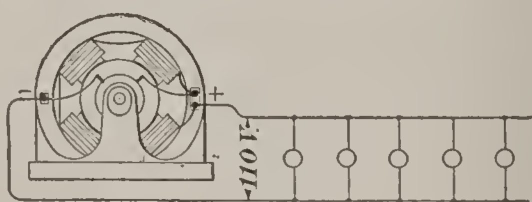


FIG. 2

for the current, such a circuit is called a *shunt*, or it is said to be in shunt with the other circuit.

Distribution Systems.—Electric circuits may also be classified as *direct-current circuits* or *alternating-current circuits*, depending on the kind of current carried. The system of conductors, or wires, leading from a power station is called a *distribution system:* it is a *direct-current system* if direct current is used, an *alternating-current system* if alternating current is used.

Direct-current is usually distributed by either the *two-wire system*, shown in Fig. 2, or the *three-wire system*, shown in Fig. 3.

Alternating current may be distributed by the *single-phase system*, the *two-phase system*, or the *three-phase system*. The single-phase system usually employs two wires, as in Fig. 2; the two-phase system, four wires; and the three-phase system, three wires. The three-phase system is similar to the direct-current three-wire system in the number of wires only. The single-phase system is rarely used for mines.

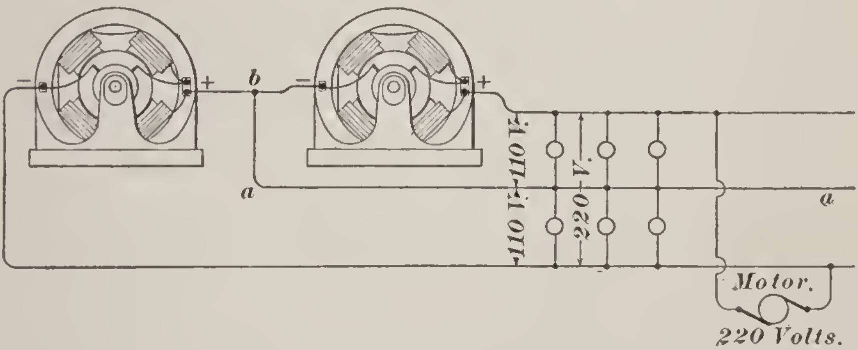


Fig. 3

Protection of Circuits.—It is necessary to protect electrical apparatus from the danger of burn-outs due to heavier currents than those for which they are designed. This is accomplished by means of *cut-outs*, which automatically open a circuit when the current exceeds a certain value. A cut-out may be either a *fuse* or a *circuit-breaker*.

Electric-Haulage Circuits.—In electric-haulage circuits, the rails take the place of one of the conductors, so that, in calculating the size of feeders required, only the overhead conductors are taken into account. It is difficult to assign any definite value to the resistance of the track circuit, as this resistance depends very largely on the quality of the rail bonding at the joints. If this bonding is well done, the resistance of the return circuit should be very low, because the cross-section of the rails is comparatively large. The following example will serve to illustrate how calculations for haulage circuits are made.

Example.—In Fig. 4 *a b* represents a section of track 4,000 ft. long. From the dynamo *c* to the beginning of the section, the distance is 1,200 ft. The trolley wire is No. 00 B. &. S., and is fed from the feeder at regular intervals. Two mining locomotives are operated, each of which takes an average current of 75 amp. The total allowable drop to the end of the line is to be 5% of the terminal voltage, which is 500 volts. Calculate the size of feeder required, assuming that the constant 14, in the formula, takes account of the resistance of the return circuit.

Solution.—As the locomotives are moving from place to place, the center of distribution for the load may be taken at the center of the 4,000 ft. The distance L will then be $1,200+2,000=3,200$ ft. The total current will be 150 amp.; hence,

$$\frac{14\times 3,200\times 150\times 100}{500\times 500}=268,800 \text{ cir. mls.}$$

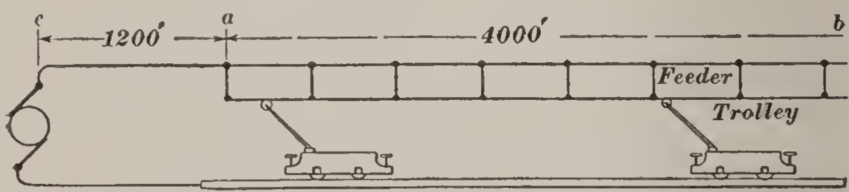


Fig. 4

This would require either a stranded cable or the use of two No. 00 wires in parallel from *c* to *a*. From *a* to *b*, the No. 00 trolley wire is in parallel with the feeder; hence, the section of feeder *a b* may be a single No. 00 wire.

In many cases, the drop is allowed to run as high as 10%, because the loads are usually heavier, and the distances longer, than in the example just given.

ELECTRIC APPARATUS IN FIREDAMP

If a fuse blows or a motor sparks in firedamp, the gas will be ignited just as surely as though it came in contact with a naked lamp. In any part of a mine where firedamp is apprehended, a safety lamp should be provided for use with each electric machine when working, and should the safety lamp give any indication of gas, the person in charge should immediately

stop the machine, cut off the current at the nearest switch, and report the matter to the proper authority. This applies especially to such apparatus as electrically driven coal cutters. The operator of electric machinery should never leave the machine running; that is, with the current on.

SIMPLE ELECTRICAL CALCULATIONS

PRACTICAL UNITS

In electrical work it is necessary to have units in terms of which to express the different quantities entering into calculations. The four most important of these are used to express current, electrical pressure, or electromotive force, resistance, and power.

Current.—The *current* in a wire may be indicated in several ways. If a compass needle is held under or over a wire, it will be deflected and will tend to stand at right angles to the wire. The stronger the current, the greater will be the deflection of the needle. The unit used to express current is called the *ampere.* The expression of current through a wire as so many amperes is analogous to the expression of the flow of water through a pipe as so many gallons per second.

Electromotive Force.—In order that a current may pass through a wire, there must be an electrical pressure of some kind to cause the flow, just as in hydraulics there must always be a head or pressure before water can be made to flow through a pipe. Howcver, there may be a pressure or head without there being any flow of water, because the opening in the pipe might be closed, though as soon as the valve closing the pipe is opened, the current will start. In the same way, an electrical pressure or *electromotive force* (usually written E. M. F.) may exist in a circuit, but no current can pass until the circuit is closed or until the wire is connected so that there will be a path for the current. The practical unit of electromotive forcc is the *volt.* It is thc unit of electrical pressure, and fulfills somewhat the same purpose as pounds per square inch in hydraulic and stcam engineering.

MECHANICAL EQUIVALENTS OF ELECTRICAL UNITS

Unit	Kind of Unit	Mechanical Equivalents	Unit	Kind of Unit	Mechanical Equivalents
1 Watt..	Rate of doing work	1 amp. at 1 volt .7373 ft.-lb. per sec. 44.238 ft.-lb. per min. 2,654.28 ft.-lb. per hr. .00134 H. P. $\frac{1}{746}$ H. P.	1 Watt-hr.	Quantity of work	2,654.28 ft.-lb. 1 amp.-hr.$\times$1 volt .00134 H. P.-hr. $\frac{1}{746}$ H. P.-hr.
1 K. W..	Rate of doing work	737.3 ft.-lb. per sec. 44,238 ft.-lb. per min. 1.34 H. P.	1 H. P.-hr.	Quantity of work	1,980,000 ft.-lb. 375 mi-lb. 746 watt-hr. .746 K. W.-hr.
1 H. P...	Rate of doing work	550 ft.-lb. per sec. 33,000 ft.-lb. per min. 375 mi.-lb. per hr. 746 watts .746 K. W.	1 Amp.-hr.	Quantity of electricity	1 amp. for 1 hr. irrespective of voltage
			Torque....	Force moving in a circle	A force of 1 lb. at a radius of 1 ft.

Resistance.—All conductors offer more or less resistance to a current of electricity, just as water encounters friction in passing through a pipe. The amount of this resistance depends on the length of the wire, the diameter of the wire, and the material of which the wire is composed. The resistance of all metals also increases with the temperature. The practical unit of resistance is the *ohm.* A conductor has a resistance of 1 ohm when the pressure required to set up 1 ampere through it is 1 volt. In other words, the *drop*, or fall, in pressure through a resistance of 1 ohm, when a current of 1 amp. is passing, is 1 volt. 1,000 ft. of copper wire .1 in. in diameter has a resistance of nearly 1 ohm at ordinary temperatures.

Power.—The electrical unit of power is the *watt.* It is equal to the power developed by a current of 1 amp. under a pressure of 1 volt. The watt is equal to $\frac{1}{746}$ H. P. The watt is too small a unit for convenient use in many cases, so that the kilowatt, equal to 1,000 watts, is frequently used. The word kilowatt is sometimes abbreviated to kw. or K. W.

Work.—The electrical unit of work is the *watt-hour.* This is the total work done at the rate of 1 watt for 1 hr. A *kilowatt hour* is 1,000 watt-hours. It is equivalent to $\frac{1000}{746}$, or about $1\frac{1}{3}$ H. P.-hr.

OHM'S LAW

Ohm's law may be briefly stated as follows:

Law.—*The strength of the current in any circuit is directly proportional to the E. M. F. in the circuit, and inversely proportional to the resistance of the circuit.*

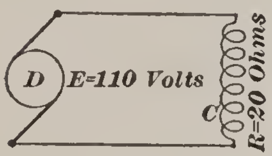
This means that if the resistance of a circuit is fixed, and the E. M. F. varied, the current will be doubled if the E. M. F. is doubled. Also, if the E. M. F. is fixed, and the resistance doubled, the current will be halved.

Let E = E. M. F., in volts;
R = resistance, in ohms;
I = current, in amperes.

Then, $$I=\frac{E}{R}, \text{ or } R=\frac{E}{I}, \text{ or } E=IR$$

The last two forms are useful in many cases where the usual form $I=\frac{E}{R}$ is not directly applicable.

EXAMPLE 1.—In the accompanying figure, a dynamo D that generates 110 volts, is connected to a coil or wire C, which has a resistance of 20 ohms; what current will pass, supposing the resistance of the rest of the circuit to be negligible?



SOLUTION.—Here, $E=110$ volts; $R=20$ ohms; hence, $I=\frac{110}{20}=5.5$ amp.

EXAMPLE 2.—If the resistance of the coil C is 6 ohms, what E. M. F. must the dynamo generate in order to set up a current of 15 amp. through it?

SOLUTION.—The third form of the law given is more convenient in this case.

$$E=15\times 6=90 \text{ volts}$$

In case the current and E. M. F. are known, the resistance of the circuit may be calculated by using the second form of the law given above. For example, if the current in the example just given were 8 amp. and the E. M. F. of the dynamo 110 volts, the resistance of the circuit must be

$$R=\frac{110}{8}=13.75 \text{ ohms}$$

POWER CALCULATIONS

Power in Direct-Current Circuits.—The power in any direct-current circuit may be found by multiplying the current by the pressure required to maintain the current in the circuit.

Let W = power, in watts;

H. P. = horsepower.

Then, watts = volts × amperes, or

$$W=E\,I$$

$$W=I^2\,R$$

$$\text{H. P.}=\frac{W}{746}$$

Power in Alternating-Current Circuits.—Owing to the nature of alternating currents the formula $W=E\,I$ cannot always be used for alternating-current calculations. It may be used for single-phase circuits to which nothing but incandescent

lamps are connected. On single-phase circuits operating motors only the formula becomes approximately $W = .8\ E\ I$, and on single-phase circuits containing both lamps and motors the formula $W = .85\ E\ I$ may be used. For two-phase circuits the foregoing expressions will give the power for one phase, so that to obtain the total power for both phases multiply by 2. On three-phase circuits, the expressions for power are $W = 1.732\ E\ I$, $W = 1.386\ E\ I$, and $W = 1.472\ E\ I$, respectively, for the three conditions mentioned.

WIRE DATA

Estimation of Cross-Section of Wires.—The diameter of round wires is usually given in the tables in decimals of an inch, and the area of cross-section is given in terms of a unit called circular mil. This is done simply for convenience in calculation, as it makes calculations of the cross-section much simpler than if the square inch were used as the unit area. A *mil* is $\frac{1}{1000}$ in., or .001 in. A *circular mil* is the area of a circle, the diameter of which is $\frac{1}{1000}$ in., or 1 mil, or .0000007854 sq. in.

If the diameter of the conductor were 1 in., its area would be .7854 sq. in., and the number of circular mils in its area would be $.7854 \div .0000007854 = 1{,}000{,}000$; but 1 in. = 1,000 mils, and $(1{,}000)^2 = 1{,}000{,}000$; hence, the following is true: $C.\ M. = d^2$; or the area of cross-section of a wire, in circular mils, is equal to the square of its diameter expressed in mils.

Example.—If a wire has a diameter of .101 in., what is its area in circular mils?

Solution.— .101 in. = 101 mils; hence, $C.\ M. = (101)^2 = 10{,}201$.

Estimation of Resistance.—The resistance of any conductor is directly proportional to its length, and inversely proportional to its area of cross-section, or

$$R = K\frac{L}{A},$$

in which R = resistance;
L = length;
A = area of cross-section;
K = constant.

If L is expressed in feet and A is expressed in circular mils, the constant K must be the resistance of 1 ft. of the wire in question of 1 cir. mil cross-section. The resistance of 1 mil-ft. of copper wire at 75° F. is about 10.8 ohms. Hence, for copper wire, $R=\frac{10.8L}{A}$; but $A=d^2$ when d is the diameter in mils; hence,

$$R=\frac{10.8L}{d^2}$$

This formula is easily remembered, and is very convenient for estimating the resistance of any length of wire of given diameter when a wire table is not at hand, or when the diameter of the given wire does not correspond to anything given in the table.

EXAMPLE.—Find the resistance of 1 mi. of copper wire .20 in. in diameter.

SOLUTION.— 1 mi. = 5,280 ft., .20 in. = 200 mils. Area of cross-section = $(200)^2$ = 40,000 cir. mils. Hence,

$$R=\frac{10.8\times 5{,}280}{40{,}000}=1.42 \text{ ohms}$$

Wire Gauge.—The gauge most generally used in America to designate the different sizes of copper wire is the American, or Brown & Sharpe (B. &. S.). The sizes given by this gauge range from No. 0000, the largest, .460 in. diameter, to No. 40, the finest, .003 in. diameter. Wire drawn to the sizes given by this gauge is always more readily obtained than sizes according to other gauges; hence, in selecting line wire for any purpose it is always desirable, if possible, to give the size required as a wire of the B. &. S. gauge. A wire can usually be selected from this gauge, which will be suited to ordinary wiring requirements in and about mines.

In the accompanying table, the common sizes of wire are listed. Wires smaller than No. 14 are not given, because No. 14 is the smallest size ordinarily used for light wiring. Trolley wires are usually made of either No. 00 or No. 0000. Conductors larger than No. 0000 are stranded and have no gauge number. Stranded conductors, called *cables*, are specified by their area in circular mils.

The table gives the dimensions, weight, and resistance of pure copper wire. The weights given are for bare wire. The first column gives the B. &. S. gauge number, the second the diameter in mils; the diameter in inches is the number given

PROPERTIES OF COPPER WIRE

B. & S. Gauge Number	Diameter Mils.	Area C. M. $=d^2$ Circular Mils.	Weight per 1,000 Ft. Pounds	Weight per Mile Pounds	Resistance per 1,000 Ft. at 68° F. International Ohms	Current Capacity National Board Fire Underwriters (Amperes)	
						Weather-Proof	Rubber-Covered
0000	460.0	211,600.0	640.50	3,381.4	.0489	312	210
000	409.6	167,805.0	508.00	2,682.2	.0617	262	177
00	364.8	133,079.4	402.80	2,126.8	.0778	220	150
0	324.9	105,534.5	319.50	1,686.9	.0981	185	127
1	289.3	83,694.2	253.30	1,337.2	.1237	156	107
2	257.6	66,373.0	200.90	1,060.6	.1560	131	90
3	229.4	52,634.0	159.30	841.1	.1967	110	76
4	204.3	41,742.0	126.40	667.4	.2480	92	65
5	181.9	33,102.0	100.20	529.1	.3128	77	54
6	162.0	26,250.5	79.46	419.5	.3944	65	46
7	144.3	20,816.0	63.02	332.7	.4973		
8	128.5	16,509.0	49.98	263.9	.6271	46	33
9	114.4	13,094.0	39.63	209.2	.7908		
10	101.8	10,381.0	31.43	165.9	.9972	32	24
11	90.7	8,234.0	24.93	131.6	1.2570		
12	80.8	6,529.9	19.77	104.4	1,5860	23	17
13	71.9	5,178.4	15.68	82.8	1.9990		
14	64.1	4,106.8	12.43	76.2	2.5210	16	12

in this column, divided by 1,000. The third column gives the area in circular mils, the numbers in this column being equal to the squares of those in the second column. The safe carrying capacity is also given.

Aluminum Wire.—The resistance of aluminum wire is practically $1\frac{2}{3}$ times that of copper wire. For insulated (covered) aluminum wire, the safe-carrying capacity may be taken as 84% that of copper wire with the same kind of insulation. To estimate the resistance of aluminum wire use the formula for copper and multiply the result by $1\frac{2}{3}$. To obtain the resistances per 1,000 ft. of the various sizes, B. &. S. gauge, of aluminum wire multiply the values given in the sixth column of the table by $1\frac{2}{3}$. To obtain the carrying capacities of aluminum wires, multiply the values in the seventh or eighth column or by .84. By dividing the values of the fourth or fifth column of the table by 2 the approximate weights per 1,000 ft. or per mi. of the various sizes of aluminum wire will be obtained.

APPROXIMATE EFFICIENCIES OF DIRECT-CURRENT MOTORS

Size of of Motor Horsepower	Efficiency of Motor Per Cent.
1 to 3	80
3 to 5	82
5 to 20	85
Over 20	90

Wire Formulas.—The size of wire for a direct-current motor may be calculated by means of the formula

$$A = \frac{20{,}470 \text{ H. P. } L}{E\,e\,n},$$

in which A = the size of wire, in circular mils;
H. P. = output of motor, in horsepower;
L = distance from the motor to feeding point, in feet;
E = voltage on nameplate of motor;
e = allowable drop from feeding point to motor;
n = efficiency of motor.

This formula allows for 25% overload. The value of n is rarely known, but may be taken from the accompanying table:

The size of wire for induction motor circuits may be found by using the formula

$$A = \frac{K\text{H.P.}L}{Een}$$

in which K = constant;

and the remaining symbols have the same significance as in the formula for direct-current motors. For single-phase motors, the value of K may be taken as 47,560; for two- and three-phase motors as 23,780. Single-phase motors are rarely used in mines. The value of n for polyphase motors may be taken from the accompanying table of efficiencies.

The amperes per terminal of an induction motor is often given on the name plate, in which case the following formula may be used:

$$A = \frac{25.5CL}{e},$$

in which C = current rating; and the other letters have the meanings given before.

It is important that induction motors be operated at the voltage given on the name plate; therefore, in order to obtain the most accurate results, the value of e in the foregoing formulas should be estimated as closely as possible.

APPROXIMATE EFFICIENCIES OF INDUCTION MOTORS

Size of Motor Horsepower	Efficiency of Motor Per Cent.
Under 5	80
5 to 20	85
20 to 100	88
Over 100	90

WIRING CALCULATIONS FOR LAMPS

Incandescent-Lamp Ratings.—There are two classes of incandescent lamps in common use; the *carbon lamp* and the Mazda, or tungsten, lamp. Before the advent of the Mazda lamp, it was the practice to specify lamps by their candle-powers; as, for instance, a 16-candle power lamp or a 32-candle-power lamp, abbreviated to 16-c.p. or 32-c.p. Now, however, lamps are rated according to the watts required to operate them. A 16-c.p. carbon lamp is now known as a 50-watt lamp; a 32-c.p. lamp becomes a 100-watt lamp. The Mazda lamps most commonly used are the 25-, 40-, 60-, and 100-watt sizes. Most incandescent lamps are made for voltages ranging from 100 to 130, although lamps designed for other voltages may be obtained.

Formulas for Lamp Wiring.—Because, with slight modifications, it may be adapted to any of the common distribution

systems, the most convenient formula for lamp-wiring calculations is

$$A = \frac{KWL}{Ee},$$

in which A = size of wire, in circular mils;
K = constant;
W = power required by all lamps of group to which wiring is run, in watts;
L = distance from feeding point to lamps, in feet;
E = voltage of circuit at lamps;
e = allowable drop in circuit.

For direct-current, two-wire and three-wire circuits and for single-phase, alternating current, two-wire and three-wire circuits, the value of K is 22. For two-phase four-wire, and three-phase three-wire circuits, the value of K is 11. On a two-phase or three-phase circuit, the lamps should be, as far as possible, distributed equally; that is, each phase should have the same number of lamps connected to it.

If all the lamps are of equal size, $W = Nw$, where N is the number of lamps and w is the watts per lamp. The formula just given then becomes

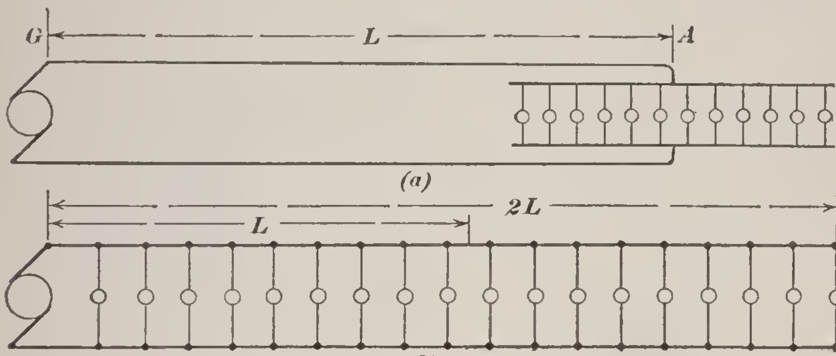
$$A = \frac{KNwL}{Ee}$$

If a certain size of wire is at hand and it is desired to know the number of lamps that the given size of wire will feed with a certain drop, the last formula may be used in this manner:

$$N = \frac{AEe}{KwL}$$

The length to be used in the wiring formula is the average distance traversed by the current in the conductor. For example, if, as in view (a) the lamps were all grouped or bunched at the end of the line, the length used in the formula would be that from G to A. In case the load is uniformly distributed all along the line, as shown in view (b), the current decreases step by step from the dynamo to the end. In such a case, the length or distance to be used in the formula is one-half that used in the former case, or one-half the distance from the dynamo to the end.

Arc Lamps.—Arc lamps are frequently run on constant-potential circuits, and usually consume from 400 to 500 watts. There are so many types of these lamps that it is difficult to



(a)

(b)

give any current estimates that will be generally applicable. Enclosed-arc lamps usually take from 3 to 5 amp. when run on 110-volt circuits.

ELECTRIC SIGNALING

BATTERIES

Batteries are used for various purposes in connection with mining work, principally for the operation of bells and signals. The unit of a battery is called a *cell*, a battery consisting of several cells properly connected together. An ordinary cell consists, fundamentally, of two pieces of solid conducting substances in a liquid that acts chemically upon one more than upon the other. The liquid is called the *electrolyte;* the solid substances, *electrodes.* The electrode at which the current leaves the cell is called the *cathode;* the electrode at which the current enters is called the *anode.* The anode is usually made of zinc; the cathode of carbon, copper or iron. Most cells use a chemical *depolarizer* to remove the objectional hydrogen gas that is formed by the chemical action of the electrolyte on the anode. An *open-circuit cell* is one designed for work where current is used only intermittently and the circuit is usually open. A *closed-circuit cell* is one designed for circuits that are usually closed and the current passes continuously.

For operating electric bells, any good type of open-circuit battery may be used. The Leclanché cell is largely used for this purpose, also several types of dry cells.

BELL WIRING

The simple bell circuit is shown in Fig. 1, where *p* is the push button, *b* the bell, and *c* the cells of the battery connected

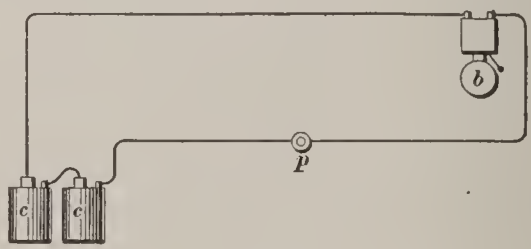


FIG. 1

up in series. When two or more bells are to be rung from one push button, they may be joined up in parallel across the

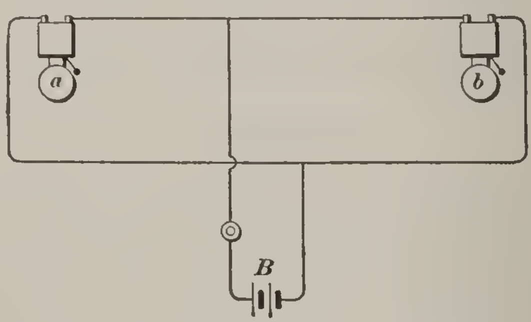


FIG. 2

battery wires, as at *a* and *b*, Fig. 2, or they may be arranged in series, as in Fig. 3. The battery *B* is indicated in each dia-

FIG. 3

gram by short parallel lines, this being the conventional method. In the parallel arrangement, the bells are independent of each

other, and the failure of one to ring would not affect the other; but in the series grouping, all but one bell must be changed to a single-stroke action, so that each impulse of current will produce only one movement of the hammer. The current is then interrupted by the vibrator in the remaining bell, the result being that each bell will ring with full power. The only change necessary to produce this effect is to cut out the circuit-breaker on all but one bell by connecting the ends of the magnet wires directly to the terminals of the bells in the circuit.

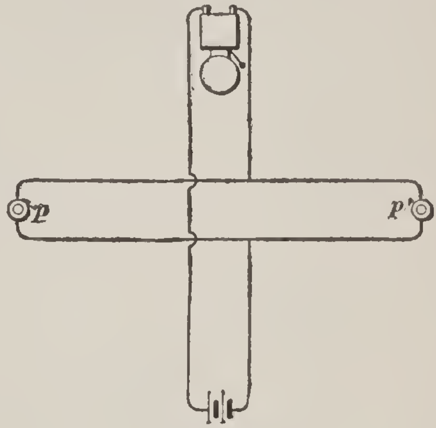


Fig. 4

When it is desired to ring a bell from one of two places some distance apart, the wires may be run as shown in Fig. 4. The pushes *p* are located at the required points, and the battery and bell are put in series with each other across the wires joining the pushes.

A single wire may be used to ring signal bells at each end of a line, the connections then being as shown in Fig. 5. Two batteries *B* and *B'*, and a key and bell at each station are required. The keys are of the double-contact type, making connections normally between the bell and line wire *L*. When one key is depressed, a current from the battery *B* passes along

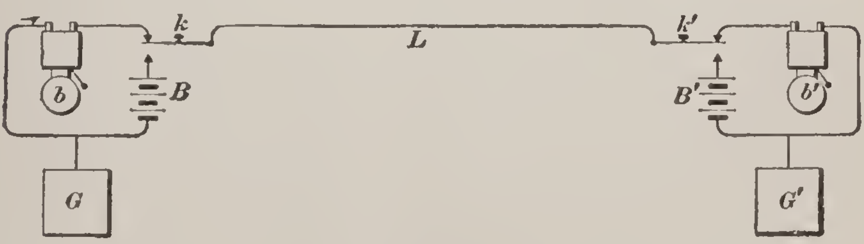


Fig. 5

the wire through the upper contact of key *k'* to bell *b'* and back through ground plates *G'* and *G*.

When a bell is intended for use as an alarm apparatus, a constant-ringing attachment may be introduced, which closes the bell circuit through an extra wire as soon as the trip at door or window is disturbed. In the diagram, Fig. 6, the main circuit, when the push *p* is depressed, is through the automatic drop *d* by way of the terminals *a* and *b* to the bell and battery. This current releases a pivoted arm which, on falling, completes the circuit between *b* and *c*, establishing a new path for the current by way of *e*, independent of the push *p*.

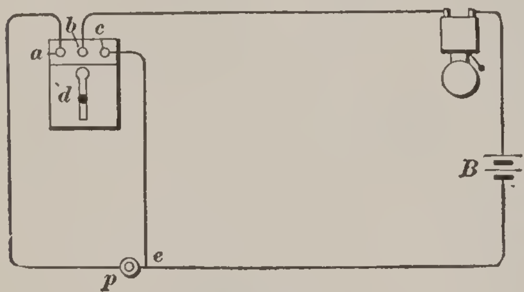


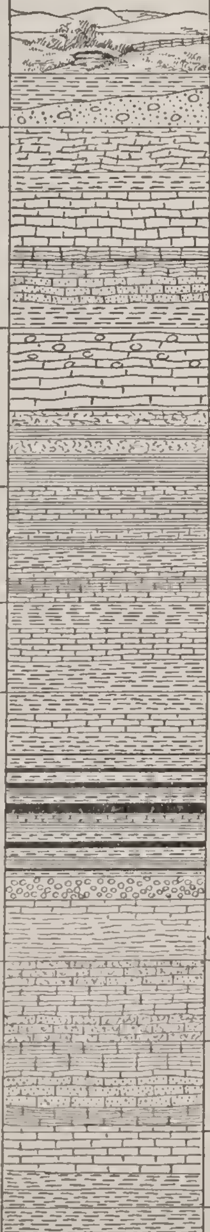
Fig. 6

The failure of a bell to operate is usually due to one of the following causes: Break in the circuit, crossed wires, wrong connections, weak battery, wrong adjustment of the vibrator. Inspection is necessary to locate the fault. If the bell rings weakly, it is usually due to poor insulation somewhere on the circuit or to a weak battery. Probably the most convenient way to test a battery for weakness is to connect a good bell directly across the battery terminals.

PROSPECTING

Formations Likely to Contain Coal.—No coal beds of importance have as yet been found below the Carboniferous period. Though coal may be looked for in any later stratified or sedimentary rocks the bulk of the best coal has been found in deposits made during this period. As a rule, highly metamorphic regions and regions composed of volcanic or igneous rocks contain no coal. The rocks most common in coal measures are sandstones, limestones, shale, conglomerates, fireclays, and, in some localities, beds of iron ore.

An examination of the fossils contained in the rocks of any locality will usually determine whether the rocks belong to a



Ages	Periods	Eras	American Periods.	Foreign Equivatents.	
Genozoic	Pleistocene	Quaternary	Recent Champtain Glacial	Recent Pleistocene	Age of Man
	Neocene	Tertiary	Pliocene	Pliocene	Age of Mammals
			Miocene	Miocene	
	Eocene		Eocene	Eocene	
Mesozoic	Cretaceous	Cretaceous	(Laramie Series) Upper Cretaceous	Upper Cretaceous	Age of Reptiles
			Lower Cretaceous (Dakota Group) (Comanche Group)	Lower Cretaceous or Neocomian	
	Juratrias	Jurassic	Alantosaurus Beds	Oölite Lias	
		Triassic	Connecticut River Beds	Keuper & Rhætic Muschelkalk Bunter Sandstein	
Paleozoic	Carboniferous	Carboniferous	Permian	Permian	Age of Amphibians or Age of Acrogens (Plants of the Coal Period)
			Carboniferous or Coal Measures	Carboniferous or Coal Measures	
			Subcarboniferous	Mountain Limestone	
	Devonian	Devonian	Catskill Chemung	Old Red Sandstone	Age of Fishes
			Hamilton		
			Corniferous		
			Oriskany		

period below or above the Carboniferous, and hence whether there is a probability of the formations containing coal. Therefore, the prospector should familiarize himself with the geological periods, and with the most common fossils of the various periods. The accompanying table gives the American and foreign names of the various geological periods, together with the name of the principal form of life during each period to and including the Devonian, which is the one below the Carboniferous or coal-forming period.

Coal or Bedded Materials.—The presence of the outcrop of any bed may often be located by a terrace caused by the difference in the hardness of the strata, though any soft material overlaying a hard material will form a terrace. Usually, the outcrop of a coal terrace is accompanied by springs carrying iron in solution, which is deposited as ochery films upon the stones and vegetable matter over which the water flows. Sometimes the outcrop is characterized by a marked difference in the vegetation, for instance, the outcrop of a coal bed contained between very hard rocks will have more luxuriant vegetation than the surrounding country.

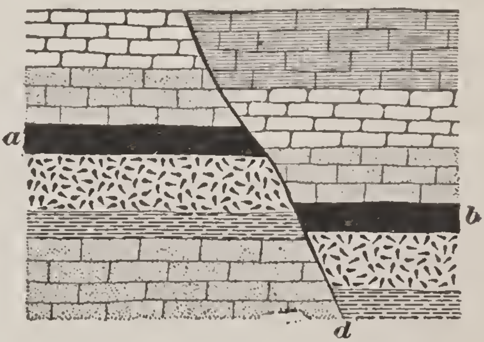
Some indication as to the dip and strike of the material composing the bed may be obtained by examining the terrace and noting the deflections from a straight line caused by the changes in contour of the ground. Where a bed or seam is faulted, its continuation can frequently be found by breaking through into the measures beyond, when an examination of the formation will indicate whether the rocks are those that usually occur above or below the desired seam.

Underground Prospecting.—Frequently a seam or deposit becomes faulted or pinched out underground, so that it is necessary to continue the search by means of underground prospecting. This work is, to a large extent, similar to surface prospecting, the underground exposures being simply additional faces for the guidance of the engineer. In the case of coal beds or similar seams, the manner of carrying on the search will depend on the character of the fault. Where sand faults or washouts are encountered, the drift or entry should be driven forwards at the angle of the seam until the continuation of the formation is encountered, when a little examination of

the rocks will indicate whether they are the underlying or overlying measures. In the case of dislocations or throws, the continuation of the bed may be looked for by *Schmidt's law of faults*, which is as follows:

Law.—*Always follow the direction of the greatest angle.*

It has been discovered that, in the majority of cases, the hanging-wall portion of the fault has moved down, and therefore such faults are commonly called *normal faults*. For instance, if the bed *a b* were being worked from *a* toward the fault, upon encountering the fault, work would be continued down on the farther side of the fault toward *d*, until the continuation of the bed toward *b* was encountered. In like manner, had the work been proceeding from *b*, the exploration would have been carried up in the direction of the greatest angle and the continuation toward *a* thus discovered. A *reverse fault* is one in which the movement has been in the opposite direction to a normal fault.



Exploration by Drilling or Bore Holes.—When testing soil or searching for placer gold, sand, soft iron, or manganese ores, and similar materials that usually occur comparatively near the surface, *earth augers*, which are very simple in construction, may be used to great advantage.

Percussion, or *churn*, *drills* are frequently used in drilling for oil, water, or gas, and were formerly much used in searching for coal and ores. They are at present little used in prospecting for either ore or coal, however, because they reduce the material passed through to small pieces or mud, and so do not produce a fair sample; besides that they can only drill perpendicular holes.

The *diamond drill* is the only form that has been universally successful in drilling in any direction through hard, soft, or variable material. Even in the use of the diamond drill, many difficulties present themselves, and demand careful study in

adapting the form of apparatus to the work in hand, and in rightly interpreting the results obtained from any set of observations. The diamond drill is used very extensively and gives the best results of any kind of drilling for exploration. The work is best done by firms that do this special work and the surveying of the bore holes must be done by the mining engineer.

MINING

OPENING A MINE

CHOOSING THE LOCATION

The location of the surface plant and the mine opening depend on the formation of the deposit and on the facilities for transporting the product to market. It is impossible for one not on the ground, and unfamiliar with natural or railroad transportation facilities in the neighborhood, to give an idea regarding the second consideration. In regard to the first consideration, the following points should be observed:

When the seam or vein outcrops within the limits of the property and is flat, a water-level drift is the best method of opening it. If the vein has any considerable inclination, it should be opened by a slope, or by a tunnel driven across the intervening measures. Where the deposit has an inclination of but from 1° to 1.25°, the water-level drift is generally used, and the main-haulage entry is opened at the lowest accessible point on the outcrop, which insures free drainage and a favorable grade for haulage. Water-level drifts, however, are only profitable where the inclined seam is exposed in ravines or gorges eroded across the strike of the measure, or where the vein can be reached by a short tunnel from the surface to the seam across the measures. This is often the case when the seam dips with the hill, but when the dip is against the hill, the tunnel is generally a long one. While the expense of operating a mine opened by a long tunnel is less than one opened by a slope or shaft, owing to cheaper drainage and haulage,

when the coal above the water level is exhausted the tunnel is almost worthless.

When the outcrop dips into the hill, the drift is usually commenced a few feet below the outcrop terrace, and is driven on a slight up grade until the normal dip is reached. When the inward dip is too strong, the better plan is to sink a shaft in the center of the basin, provided the depth is not too great and the amount of water to be pumped is comparatively small. If the inward dip to the center of the basin does not exceed a total of 25 ft. difference in level, a drift may be used and drainage effected by a siphon.

When the seam is inclined and is accessible at no point along its outcrop low enough to furnish sufficient lift or breast length, it should be opened by a slope or shaft. Or, if the seam is flat and does not crop on the tract, a shaft is the only method of working it, unless it lies so near the surface that it can be stripped.

Where a seam has a dip of 20° or more, and is brought close to the surface by an anticlinal axis or *saddle*, a *rock slope*, or, in other words, a tunnel dipping the same as the seam may be started from the surface, and, when the seam is reached, may be continued to the desired depth in the seam.

In sinking slopes for coal mines, it is customary to sink an airway alongside of and parallel with the slope, with a pillar of about 10 yd. between. The slope for coal mines is usually sunk so that there is a lift of from 100 to 110 yd., and then gangways are turned off on each side. The *lift* is the length on pitch that breasts or rooms, driven at right angles to the gangway, can be driven in good coal. Subsequent lifts are usually from 80 to 100 yd. long.

SHAFTS

Shafts and tunnels may be temporary, or those that are simply driven for exploration purposes, and are not to be used for any great length of time; or they may be permanent, or those that are driven for a specific purpose and usually have a definite predetermined capacity. In the United States, shafts are usually square or rectangular in form, as timber is used in lining them. In Europe, round or oval shafts are frequently used; these have a lining of brick, iron, concrete, or masonry.

Compartments.—The number of compartments in a shaft and their arrangement depends largely on the use to which the shaft is to be put; also on the number of shafts at the property, and the depth of the shaft. Where the material is to be removed is comparatively near the surface, it is usually cheaper to sink a number of two- or three-compartment shafts than it is to tram all the ore to one large shaft; while, in the case of very deep mines, large four- or six-compartment shafts are sunk, and the underground haulage extended over a greater area. When the shafts are lined with timber, a strong construction can be obtained by placing the compartments side by side, as shown in Fig. 1. When a body of material comparatively near the surface is being removed through a number of shafts, two-compartment shafts are frequently built, both compartments being used for hoisting, and separate shafts being provided for the pump column and ladderways. This reduces both the size of the shaft and the timbering necessary, and also does away with the special danger from fire that always exists when there is a ladderway in the shaft, for it is always difficult to fight fire in these special compartments.

Fig. 1

Size of Shafts.—Shafts vary greatly in size, depending on the number of compartments desired and the size of the compartments. For coal mines, they are generally from 10 to 12 ft. wide inside of timbers, and each compartment is from 6 to 7 ft. wide inside the guides. This would make the outside dimensions of a double-compartment shaft about 13 to 15 ft. wide, 17 to 18 ft. long, and a triple-compartment shaft from 24 to 25 ft. long.

Shaft Sinking.—As a general thing, the loose material or wash above bed rock is not thick enough to cause any serious trouble, and ordinary cribbing of heavy timber or a masonry curbing is sufficient. But when the surface is very thick or loose, and runs like quicksand, considerable difficulty is experienced. The general method of overcoming this difficulty in the past was to at once divide the shaft into the required number of compartments by heavy timbers alternating or placed skin

to skin, which had the effect of bracing the cribbing against the lateral pressure of the loose material. This method is effectual where the wash will remain solid or stand long enough to allow the timbering and cribbing to be put in. But when the surface is thick, loose, or watery, or of quicksand, some one of the following special methods of sinking must be adopted:

Quicksand................	Forepoling Metal linings, forced down without use of compressed air Pneumatic method, limited to about 100 ft. in depth Poetsch process, freezing method
Rock, hard or soft, but very wet	Kind-Chaudron method
Rock, hard or soft, but not very wet................	Continuous, or long-hole, method

When the ground is so bad that it will not stand for several days between excavation and the completion of the lining, it becomes necessary to carry the timber to the bottom of the work. This may be accomplished by using square-set shaft timbering and driving laths, or *forepoling* behind the timber so as to keep the soft material from running into the opening. The advantages of forepoling are that, if the shaft is being lined with square sets, forepoling can be commenced at any point, and, if the ground is not too bad, the work can be continued by this means until solid material is encountered. When the ground is particularly bad, it may become necessary to use *breast boards*, which are simply boards braced against the bottom of the shaft so as to keep the material from rising into the opening, only one board at a time being removed while the material behind it is excavated.

The *pneumatic method* of shaft sinking was developed from the system in use for putting down foundations for bridge piers. At the bottom of the shaft there is a small chamber called a caisson, in which a sufficient air pressure is maintained to exclude the water at all times. The shaft lining is built on above this chamber, and gradually forced down into the soil. Men enter the chamber and excavate the material from under the caisson as it descends.

By this method the sinking commences at once and is continued without interruption until the lining is completed to bed rock, to which the lining is joined, as shown in Fig. 2. An air compressor, which is subsequently used, is the only auxiliary machine necessary.

In the *pneumatic process*, the fine material is aspirated out of the caisson by the air pressure. This process is limited to a depth of about 100 ft., as it is impossible for men to work under a greater air pressure than that which corresponds to about 100 ft. of hydrostatic pressure.

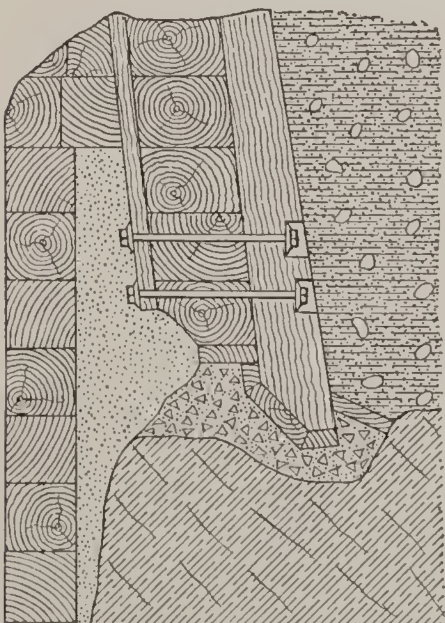


FIG. 2

By the freezing process, pipes are sunk in the ground about the area to be frozen, as a rule, not more than 3 or 4 ft. apart. The lower ends of the pipes are sealed or closed, and an inner tube introduced so that a freezing mixture may be caused to circulate down through the inner tube, and up through the outer tube. This freezing mixture may be liquid ammonia gas, which is allowed to expand in the outer tube, or a solution of calcium chloride that has been reduced to a very low temperature by means of an ordinary refrigerating machine. The circulation is maintained in the pipes until the ground between them is frozen solid, after which the work may be continued as though the formation were solid rock, the material being blasted and hoisted in buckets. The freezing process may be applied to any wet formation, whether hard or soft, while the pneumatic process is applicable only to soft formations. The freezing process may be carried to practically any depth. As a rule, the freezing pipes are never sunk inside of the shaft area.

The *Kind-Chaudron method* is applicable only to round shafts, and is suitable for shafts passing through very wet and at the same time comparatively soft formations. The excavation is carried on by means of a large set of boring tools armed with steel teeth, and operated in a manner similar to that employed in drilling wells by the percussive system.

The *long-hole process* consists in the drilling of a series of diamond-drill holes over the area of the proposed shaft, then filling the holes with sand. Afterwards 5 or 6 ft. of sand is removed from the holes in the interior of the shaft, and these holes are charged with cxplosives, and fired by electricity. Next, the holes around the boundary of the shaft are charged and fired in the same manner, and the process is continued until the bottoms of the diamond-drill holes are reached. This method is cspecially applicable to work in hard rock, where great speed in sinking is desired, for all the drilling is accomplished at one operation, after which the sinking progresses by simply cleaning out the drill hole, blasting the material and cleaning it away.

Sinking Head-Frames.—Head-frames of very simple form are used for sinking. The skeleton of the frame is formed of heavy squared timber 10 in. $\times$ 10 in. or 12 in. $\times$ 12 in. mortised and pinned together, and braced by diagonal braces. A good height from thc surface to the center of the sheave is from 20 to 25 ft. The shcave should bc from 6 to 8 ft. in diameter.

Sinking Bucket.—The sinking bucket should be of boiler iron, or of heavy hardwood strcngthened by iron bands, about 3 ft. in diameter at the top by from $2\frac{1}{2}$ to 3 ft. deep. It should be suspended by a handle pivoted a triflc below the center, and it should have a pin on the rim of the bucket that will hold it in an upright position when a loose ring on the handle is slipped over it. A chain fastened to the top of the head-frame, with a hook on its loose end, is suspended so that, when hanging plumb, it is over a chute leading to the dump car. As the bucket is hoistcd out of the shaft, this chain is attached, and the engine reversed. The buckct swings over the chute, thc ring holding it upright is knockcd off the pin, and the rock is dropped into the chutc. Rocks too large for thc bucket arc suspcnded in chains and are hoisted in that way, and removed on a truck

that runs on a track inside of the head-frame, the gauge of which is sufficiently wide to give plenty of clearance for the bucket.

Sinking Engines.—Most shafts and slopes are sunk with old engines, or else by engines especially designed for such work, and so constructed that they can easily be moved from place to place.

Tools.—The old method of hand drilling is still adhered to in many instances, but it is gradually giving way to machine drilling, especially in deep shafts. When properly managed, the work is done much more rapidly and economically by the several excellent types of rock drills now on the market. They are constructed in a variety of shapes by the makers, and there are so many convenient accessories in the shape of fittings, etc., that all contractors prominent in the various coal fields possess one or more of their favorite type of drills. These drills are run either by compressed air, steam, or electric power, and in large shafts two are usually employed, so that work may not be delayed by a breakdown of one drill.

Drainage and Ventilation.—When only a small amount of water is encountered while sinking, the best plan is to allow it to collect in a depression and bail it from there into the bucket, hoisting it the same as the rock. Where the water is excessive, in quantity, a steam pump is necessary; all the leading pump works make pumps especially designed for sinking purposes.

When the shaft is of moderate depth, a fire burning in one corner will supply ample ventilation. To rapidly clear away smoke, a good plan is to burn a bundle of straw or shavings in one end of the shaft, and throw a couple of buckets of water down the other end. When the shaft is very deep, or when the sectional area is small, ventilation is produced either by a steam jet, or by a small fan turned either by steam or by hand. In some cases, a fire is used that draws into a board pipe.

Slope Sinking.—A *slope* is an inclined plane driven down on the bed of the seam, and is generally through coal or ore, though sometimes it is driven through rock across measures to cut the seam that cannot be conveniently worked by a slope. In the latter case, it is merely an inclined tunnel; in the former it might be termed an inclined gangway. A slope and an inclined

plane, when mentioned hereafter, will mean an inclined opening in coal used as a passageway for mine cars.

When the location of the slope has been decided on, a temporary sinking plant is erected. For a short distance, varying with the nature of the ground, but usually ranging from 10 to 20 ft. on the pitch, an open cut is made, and the earth, rock, or crop coal is thrown out by hand. As soon as sufficient cover is reached, the work of undermining and timbering is commenced, and at the same time a double or single track is laid, so that the material can be taken out in a car or self-dumping skip. When the latter is used, the track is continued up a trestle some distance above the surface, and a head-sheave so placed as to draw the skip up the required distance and dump the material in a chute beneath the trestling.

The width of the slope depends on the size of the cars and the number of compartments. The most common arrangement is to divide the slope into three compartments; two large ones for hoistways, and a smaller one for pump rod, column pipe, steam pipe, and traveling way. This last is also used as an airway while sinking is going on.

The Sump.—When the shaft or slope is completed, among the first things necessary is a *sump* in which to collect the drainage of the mine. This is an opening lower in the vein, when it is a pitching one, or in the rock when it is a flat seam reached by a shaft. It should be large enough to hold any excess of water that the pumps cannot handle; and the pumping machinery should be powerful enough to handle the ordinary drainage by running not over 10 hr. per da. When this is the case, in an emergency, the pumps can be run continuously, and thus handle the surplus water.

Driving the Gangway.—In bituminous coal seams, the height of the gangway is governed by the thickness of the seam; this is also true, in a certain sense, in the anthracite regions, though in anthracite mines they are very seldom less than 6 ft. in height. In the larger seams they are from 6 ft. 6 in. to 7 ft. 6 in. high in the clear, and from 10 to 15 ft. wide. The gauge of track varies from 24 to 48 in. The grade should rise at least 4 in. in 100 ft., and a gutter 3 ft. wide by 18 in. deep should be cut in the coal on the low side.

TUNNELS

Mining tunnels are usually of small cross-section compared with those that occur in railroad work, it being rare that their size is such that they cannot be driven in full section, and if the ground is firm the operation of placing the lining may follow behind the work of driving. They are generally lined with timber, and in case the ground is of a soft or treacherous nature, bridged square sets and forepoling are employed, with or without breast boards, as the neccssity of the case demands. When the material is firm rock, the tunnel is sometimes not lined, the roof being given an arched form.

MINE TIMBER AND TIMBERING

Choice of Timber.—Timber used for underground supports in mines should be long-grained and elastic, and, at the same time, should not be too heavy. Oak, beech, and similar woods are very strong, but are heavy to handle, and when set in place are treacherous, because they are short-grained and not elastic, so that they break without warning. Mine timber is placed, not with the intention of ultimately resisting the great pressure of the earth, but to keep any loose pieces in place and to give warning to the workmen, thus enabling them to escape before a fall occurs. For this reason, pine and fir are suitable for mine timbering, as they combine a fair amount of strength with considerable elasticity, and hence give warning long before they break. Very elastic timbers, such as cypress, willow, etc., are to be avoided, for they simply bend like a bow and do not offer the necessary resistance to hold the material in place for a short time.

Preservation of Timbers.—The character of the ventilation in a mine has considerable effect on the life of any timber supports. Damp stagnant air will cause mold and fungus growth, which will be followed by the destruction of the timber through decay or dry rot. All timbered openings should be well ventilated, and provision made for the speedy removal of damp hot air, such as commonly occurs around pump rooms and along steam lines. Water is a good preservative, as it washes off the

spores of the fungi as fast as they are formed, and for this reason shaft timbers are sometimes kept wet.

Timber may be also preserved: (1) by a solution of common salt and water; (2) by impregnating the wood with such metallic substances as sulphates of copper, iron, etc.; (3) by impregnation with the chloride of magnesium or zinc; (4) by creosoting; (5) by coal tar; (6) by carbolineum.

A solution of 1 lb. of salt in 4 or 5 gal. of water gives a cheap and easily applied preservative with which the timber should be thoroughly soaked. Sulphate of iron is economical and effective. In the zinc process, a solution of 1 gal. of liquid chloride of zinc (sp. gr. 1.5) mixed with 35 gal. of water is forced into the wood by pressure, Impregnation with crude creosote oil is effective, for the creosote fills the pores and prevents saturation by water; it destroys organic life; the carbolic acid that it contains coagulates the albuminoids and prevents decay; but it has the disadvantage of making the timber very inflammable. Painting with liquid tar is effective, but makes the wood very inflammable. Painting with ordinary whitewash is also said to give good results. Carbolineum is said to be effective, but is quite expensive. It is applied with a brush, or by steeping in a tank; 1 gal. will cover 300 to 400 ft. of timber. It has been shown that preservatives decrease the strength of timber from 8% to 20%, depending on the process used.

In selecting props, the principal points to be observed are: Straightness, slowness of growth as indicated by narrow annular rings, freedom from knots, indents, resin, gum, and sap. They should also be well seasoned before use. With these precautions and proper mine ventilation, fungus growth may generally be obviated and durability insured.

Placing of Timber.—The individual sticks should never be weakened by cutting mortise and tenon joints. The pressure should be evenly distributed over a number of sticks, and not concentrated or centered at one point. Centers of revolution should be avoided. The individual sticks should be placed in the direction of the strain that they are to resist, so that they will be subject to compression along their length rather than to a transverse strain. The individual sticks should be so placed, and the joints so formed, that the pressure tends to

strengthen rather than weaken the structure up to the crushing strength of the timber. In the case of large stopes, the timbering should be done according to some regular system, while, at the face of coal mines, single props or posts are usually better, owing to the fact that their duty is only to support the loose portion of the roof for a limited time. Probably the most important point is to timber in time, before the rock becomes broken or begins to settle.

It seems generally agreed that the main weight in mines comes nearly at right angles to the bedding, and that the props should be mainly set in that direction. If the deposit is horizontal, the weight generally comes vertically; but if the deposit is inclined, the weight comes at a right angle to the inclination. Some authorities hold it as a principle that all props should be set parallel with the main pressure. Others, in order to guard against possible side thrusts and a tendency of the ordinary weight to ride to the dip in inclined deposits, purposely cause a sufficient number of props to be set with their tops slightly uphill.

Sawyer fixes a maximum and minimum slope for the props, varying with the rate of dip. He makes this maximum slope of the props one-sixth that of the dip, and the minimum slope one-third of the one-sixth.

Props are usually set with the butt end downwards, but not always. Placing the butt end upwards adds a trifle to the weight on the lower end, but the larger size at the top lessens the liability of its being split by a coupling resting on it, and also gives more surface for abrasion in hammering up against a rough roof.

Joints in Mine Timbering.—In all mine timbering, the object is to so form the joints that no fastenings will be necessary and that the pressure from the surrounding material will keep the joints tight. The reason for this is that metal joints usually corrode rapidly in mines, while the timbering can be replaced with greater ease if the sticks are so framed that, by relieving them temporarily of the pressure from the sides and top, they can be simply lifted out of place and new ones substituted. The use of a framing machine renders it possible to frame the joints more exactly than with hand framing. With hand-framed

timbers, the joints are always cut a little free to allow for any unevenness in the surface, but, if machine-framed, they are sure to be of the same size. As timber does not shrink in the direction of its grain, if the caps shrink slightly, they will become loose in the space between the shoulders; hence, if timbers are cut green and framed to the exact size, subsequent shrinking may open some of the joints. This may be obviated by keeping the timber moist.

Undersetting of Props.—Props at the working face should not be set at right angles to the inclined floor of the seam, but should be underset, and the greater the inclination, the greater the underset. The amount of underset should vary with the inclination of the seam, and should not be so great that the props will fall out before the roof has tightened them.

Forms of Mine Timbering and Underground Supports.—The timbering of a mine may be divided into two heads: timbering the working faces and timbering the roads. The roof may be supported (1) by packing the waste places entirely where sufficient material is obtainable for the purpose, and timbering the face and roads; (2) by partially packing the waste, by cribs or stone pillars with intervening spaces, and by timbering the face and roads; (3) by timbering the face and roads and supporting the roof in the waste places by wooden or stone pillars, but without any packing; (4) by timbering alone without any packs or walls whatever; (5) by supporting the main roads with brick arching, or by steel or iron supports. The accompanying figures show a number of the common forms of mine timbering and underground supports.

Fig. 1 shows a *post a* and *breast cap b*. The breast cap *b* is also sometimes called *cap*, *head-block*, *headboard*, *lid* or *bonnet*. Sometimes the posts are placed upon blocks of wood similar to the head-blocks or headboards, the block being called a *sole;* at other times, two or more posts may be set upon one long block of timber called a *sill*. When posts are used in inclines, they should not be set perpendicular to the foot and hanging walls, but should be underset slightly, so that any tendency of the hanging wall to settle will bring the posts nearer at right angles to the walls, and so tighten them; the amount of underset should never be more than one-sixth the pitch of the deposit.

Where posts are set at an angle, they are usually placed on wedges, and, as the pressure comes on, the wedges are tightened.

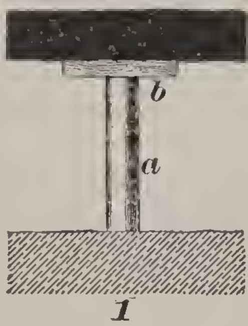


FIG. 1

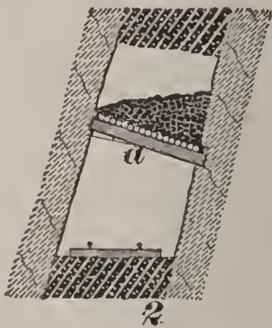


FIG. 2

Fig. 2 represents a *stull a*, which is used either to keep the walls of perpendicular or steeply inclined beds or veins apart, to support planking or lagging as a working platform, or as a platform upon which to pile ore or rock.

Fig. 3 represents *cockermegs*, which are simply timber frames used in coal mines for holding the face of the coal in place while it is being undercut. They are composed of a pole *c* extending along the face and supported by short stulls or braces *a*, the whole being tightened into place by the long stulls *b*.

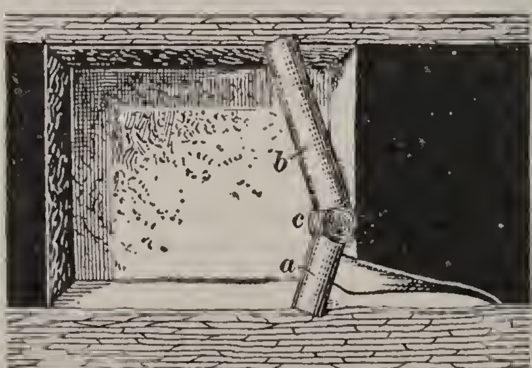


FIG. 3

Fig. 4 shows a *crib*, *cog*, *chock*, *pillar*, or *shanty* built up of timbers and filled with waste rock. It is intended to serve as a

pillar and to withstand great vertical pressure, doing away sometimes with the necessity of leaving pillars of ore. Fig. 5

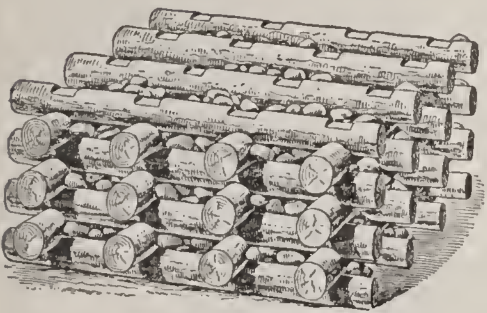


FIG. 4

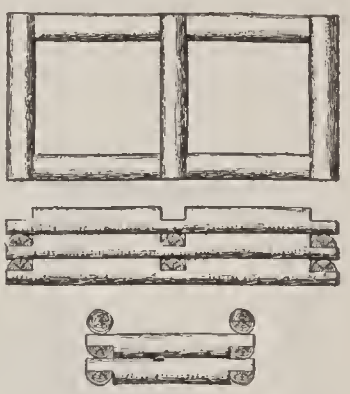


FIG. 5

is a *cribbing* framed from round timbers laid skin to skin, and used in raises or ore chutes.

Gangway or Level Timbers.—Fig. 6 is a set used in the case of an extra-wide gangway, there being a center post under the middle of the cap. This form of set may be provided with a sill when the floor of the drift or gangway is soft. Fig. 7 shows a form of drift set surrounded by bridging and used where such bad ground is encountered as to necessitate forepoling. At *A* are shown the posts, *B* the caps, and *C* the sill of the regular

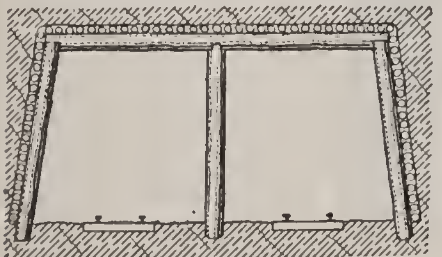


FIG. 6

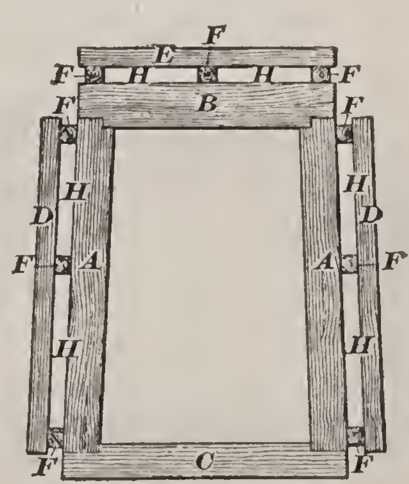


FIG. 7

set; *D* are upright bridge pieces; *E* a horizontal bridge piece separated from the set proper by blocks *F* so as to provide

spaces H around the regular set through which the spiles or forepoles can be driven. Fig. 8 shows a form of drift set sometimes used in very heavy or swelling ground. This method

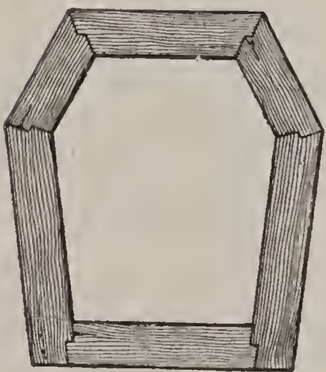


FIG. 8

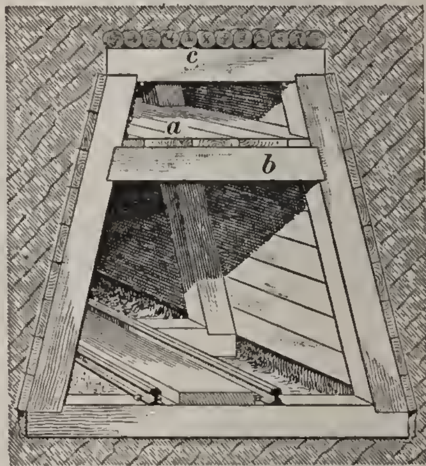


FIG. 9

of framing the timbers shortens each piece and reduces the transverse strain on all the timbers.

Fig. 9 shows an ordinary drift set provided with a sollar for ventilation purposes. An additional brace b is placed parallel to the cap c, and this is covered with plank lagging a, so as to provide a passage above the regular drift, which may be used as a return air-course.

FIG. 10

Fig. 10 is a simple form of drift set used when the roof and walls are of soft material, but the floor material firm. It is composed of posts l, upon which is placed the cap c. The joggle cut into the cap to receive the heads of the post should never be less than 1 in. nor more than one-third the thickness of the cap. The cap is usually made of such a length that the posts l have

an inclination or batter as shown in the illustration, thus giving greater strength to resist side pressure without decreasing the floor area of the drift, which may be necessary for drains, ditches, water pipes, etc. at the sides of the track. When the floor is not composed of solid material, the posts *l* may be set upon a sill that is framed to fit the legs in a manner similar to that shown for the cap. The joggle cut in the sill should never be less than 1 in. nor more than one-third the thickness of the sill. The sill is usually composed of lighter material than the cap, is flattened on one or both sides, and is sometimes used as one of the ties to receive the track.

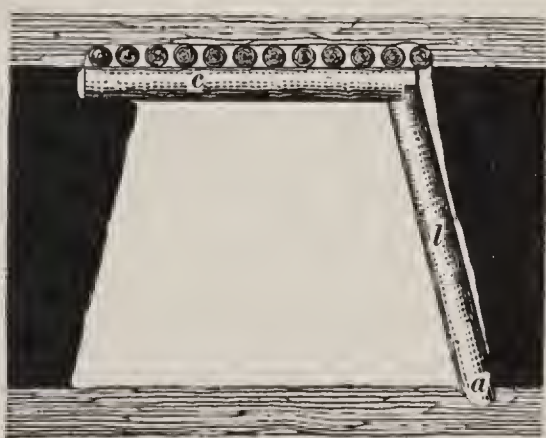


FIG. 11

Fig. 11 shows a *post l* and the cap or *collar c*, used where one wall is of firm material. On one end the cap is placed in a hitch. When the collar is supported in a hitch, it is sometimes said to be *needled*, the operation being called *needling*. The bottom of the post *a* is also secured in a hitch, in case there is any side pressure. To keep the surrounding material in place, lagging is necessary, as shown behind the timbers in Figs. 10 and 11. In the case of running ground, the lagging is usually made from sawed material and driven close together.

Fig. 12 illustrates a method of *spiling* or *forepoling; a* are the posts of the regular set, *b* the caps, and *e* the top bridging. The front ends of the spiles from any given set rest on the bridging

of the next advanced set, and the spiles for advancing the work are driven between the bridging and the set, as shown in the illustration. To force the spiles out into the ground, so as to provide room for the placing of the next set, *tail-pieces i* are placed behind the back end of the spiles as they are being driven. After the spiles have been driven forwards the desired amount, another set is placed, the tail-pieces knocked out, and the front end of the spiles allowed to settle against the bridging of a new set. Where the face is composed of extremely bad material, it may be necessary to hold it in place with *breast boards k*, which are held in place by props *l*, that rest against the forward set. When breast boards are used, it is usually necessary to employ foot and collar braces between the sets, so as to transfer the pressure of the breast back through several sets.

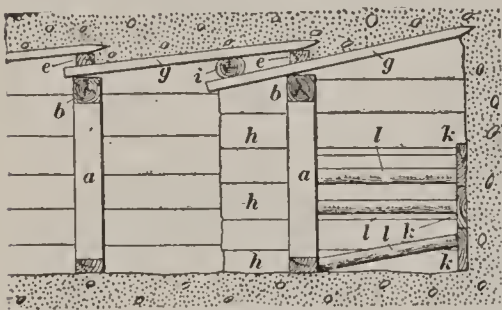


FIG. 12

Fig. 13 shows a method of placing drift sets in the case of very heavy or swelling ground; *a* are the posts, *b* the caps, *c* the sills, *d* the *collar braces* that bear against both the caps and the posts, *e* the *foot* or *heel braces* that bear against both the sills and the posts, *f* the *diagonal braces* that are halved together and placed as shown.

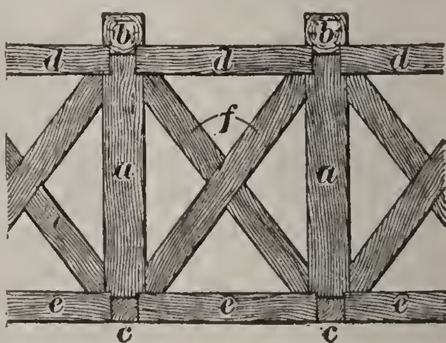


FIG. 13

Shaft Timbering.—Fig. 14 shows square-set timbering, sometimes used for shaft lining; *A* are the wall plates, *B* the end plates, *C* the buntons, and *D* the posts. The method of framing the different parts is plainly shown.

Fig. 15 represents *cribbing* sometimes used for shafts. It is composed of heavy sawed material halved together at the ends,

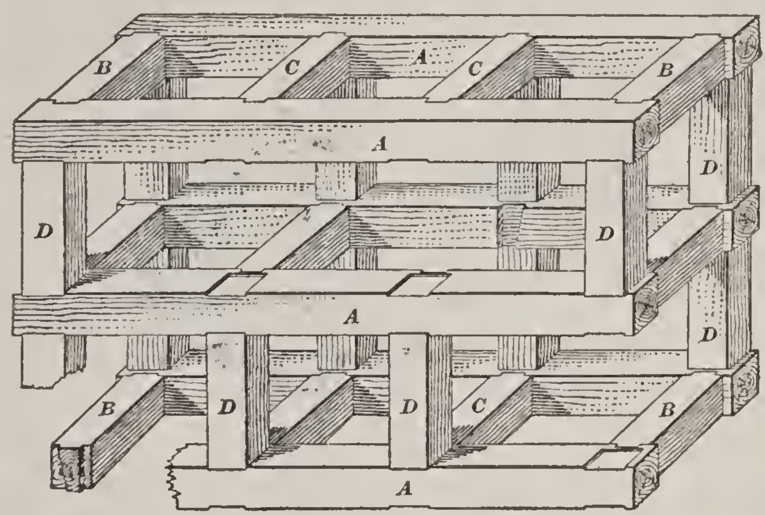


FIG. 14

as shown. The long pieces *a* are called *wall plates*, and the short pieces *b*, *end plates*. Between the compartments a partition is built up of pieces *c* called *buntons*. The ends of the buntons are let into the wall plates an inch or so, and should be so placed that they will break joints with the individual pieces of the wall plates, thus preventing the timbers of any single set from bulging into the shaft.

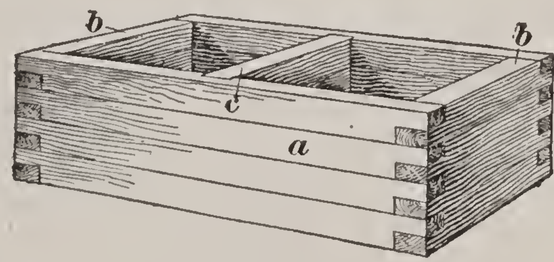


FIG. 15

Fig. 16 shows another method of framing, sometimes used for the end and wall plates where square-set timbering is used

in shafts. The end and wall plates are halved together as shown. A beveled face is often formed at *D*. This construction necessitates the cutting of a tenon on the end of the post *F*

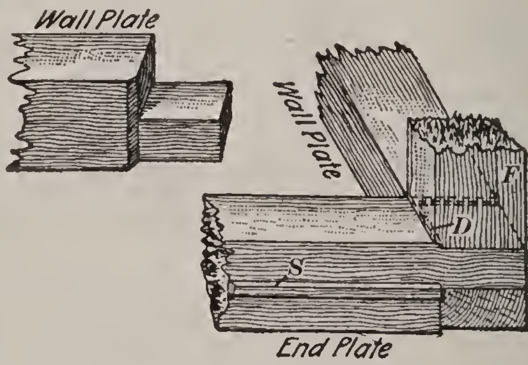


FIG. 16

as shown; *S* is a 2 in.×2 in. strip nailed along the center of the back of the wall and end plates as a support for the lagging that is placed outside of the sets. The lagging is usually composed of 2 in.×3 in. plank.

Fig. 17 shows the use of *hangers* between the individual square sets. The hangers are bolts provided with hooks on the ends, and are used to support the sets as the work progresses, the posts serving to keep the sets properly spaced, while the hangers keep the sets tight against the posts. Hangers are not always left in permanently, but may be removed after a considerable section of the shaft has been completed.

FIG. 17

Fig. 18 shows a method of applying rough square sets, made from round timber, to the sinking of a small prospecting shaft by the use of forepoling; *A* is the first set of timbers and *J* is the second. The hangers are made from 2 in.×4 in. timbers *F* spiked to the sets and to the supports *G*. The supports *G* from which the sets are hung are placed over sills *H*, which are situated at a convenient distance from the collar of the shaft.

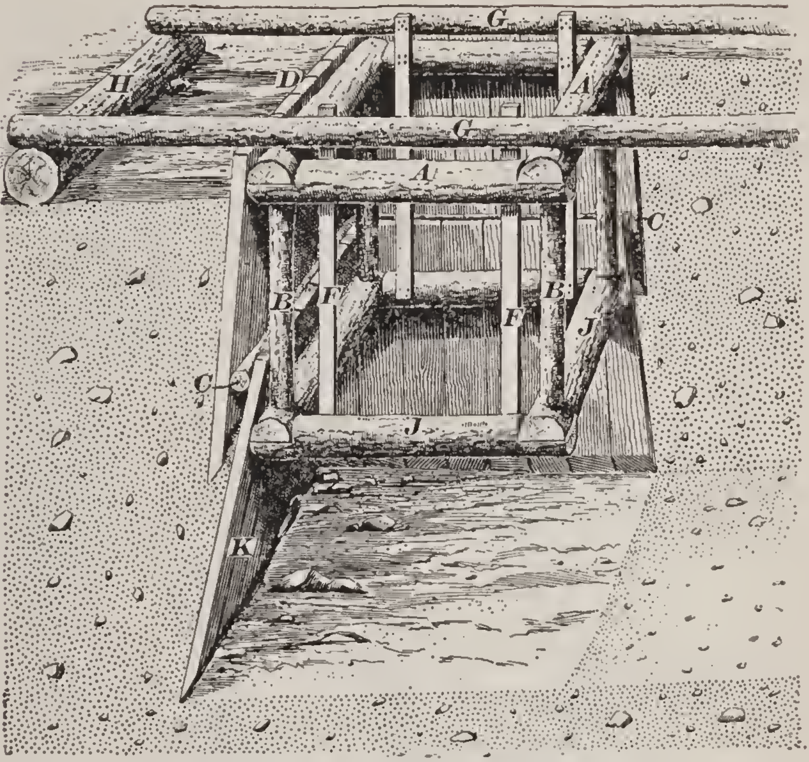


Fig. 18

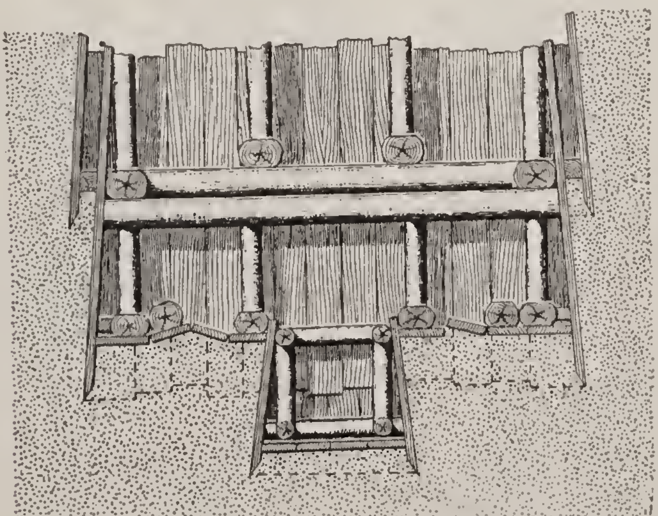


Fig. 19

At D is shown the lagging of the first set that is usually spiked to the set and at K the forepoling, which becomes the lagging between the second and third sets, and C the tail-pieces used for forcing the lagging out into the ground. The hangers between the next two sets would be spiked to the other two timbers of the sets. Where the bottom of the shaft is very bad, it may be necessary to use breast boards, as shown in Fig. 19, in which the shaft is being put down by means of square sets and forepoling with the use of breast boards.

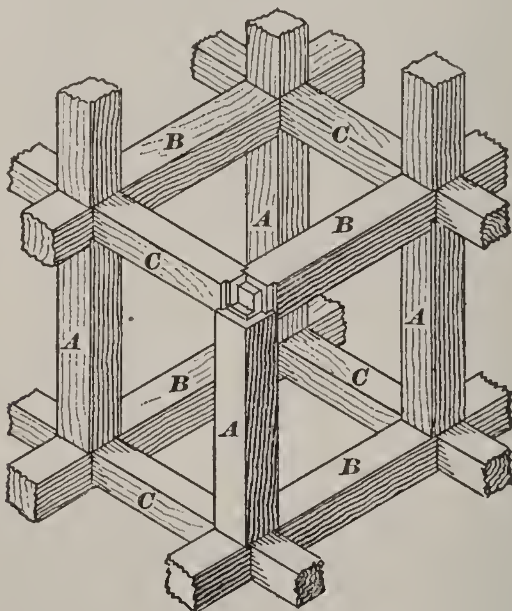


Fig. 20

Square Sets.—Fig. 20 illustrates one method of framing square-set timbers from sawed material for use in stopes in mines; A are the posts, B the caps and sills, while C are the *sprags* or *stulls*. The method of framing the joints is clearly shown in the illustration. Sometimes both caps and sprags are made of the same sized material and are framed alike. Fig. 21 shows a method of framing round timbers for square sets. The dimensions f and c are usually made about 10 in., d, e, and i, each 2 in.; a depends on the diameter of the post that is to be used; and b as a general rule is cut down to an angle of about 45°.

Landings, Plats, or Stations.—Fig. 22 is one method of timbering a plat or station. The regular square-set timbering of the shaft is continued past the station and the

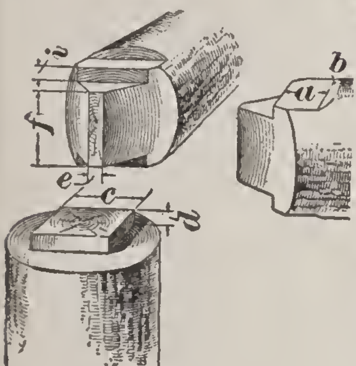


FIG. 21

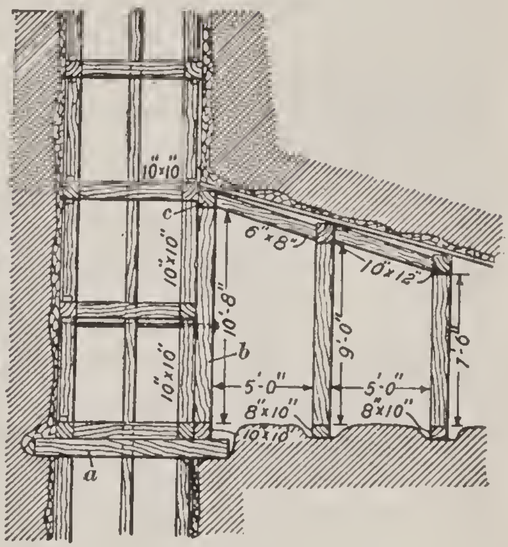


FIG. 22

heavy *stull* or *reacher* *a* put across at the bottom of the station. The posts *b* are bolted against the posts of the sets and the

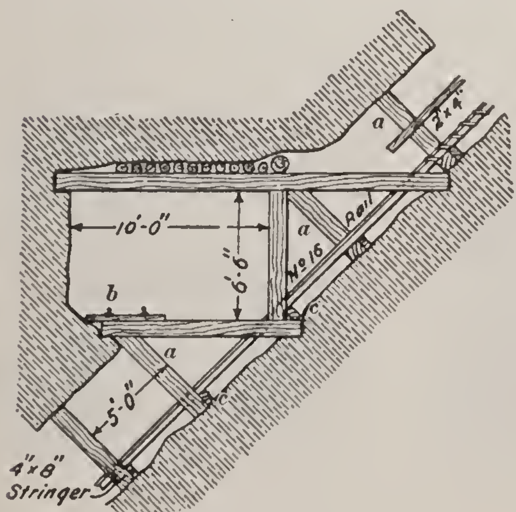


FIG. 23

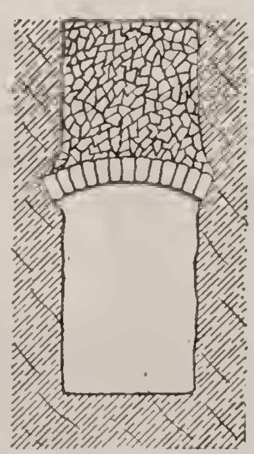


FIG. 24

cap *c* placed on top of them. After this the wall plates are cut out between the posts *b*, and the station opened and

timbered as shown. The height of the station is gradually reduced to that of the drift or level connecting with it.

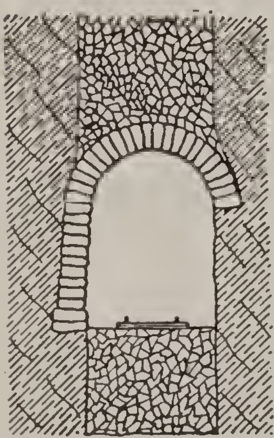


Fig. 25

Fig. 23 shows a method of timbering a level in a slope where the ground is so firm that only stulls are used in the slope and at the station, the timbers all being secured in hitches or by stulls. Here *a* represents the stulls and *c* the timbers that are spiked to the stulls and carry the stringers for the car track; *b* represents the car track from the level that is brought across above the skip track.

Special Forms of Supports.—Fig. 24 shows a stone arch that, as a stull, supports the waste material in the level. Fig. 25 shows a stone arch when one wall of the formation requires support. Fig. 26 illustrates a passage lined by a combination of stone or brick walls with wooden caps and lagging for the roof. Fig. 27 illustrates the lining of a drift or level supported by means of iron or steel shapes bent into the form of an arch and used for the support of lagging.

Fig. 28 illustrates a cast-iron post or stull that has been successfully used as a support in mines. It is composed of two

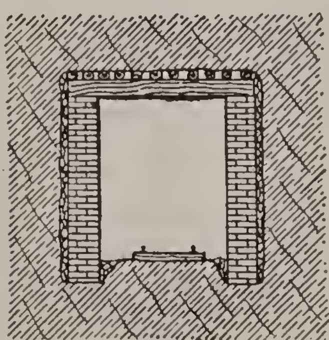


Fig. 26

Fig. 27

pieces *a* and *b*, held together by a collar *c*. By driving the collar *c* down on the post, the two pieces can be taken apart and the post moved.

Fig. 29 illustrates a masonry shaft lining supported by means of cast-iron plates *C* set in bell-shaped cavities cut in the walls of the shaft. As the masonry of that section from below is built up toward that above, the overhanging portion *D* is cut out a little at a time, and the masonry from below built up under the plate so that the lining becomes continuous.

Fig. 28

Fig. 30 illustrates masonry shaft linings, supported by artificial stone or cement foundations built in bell-shaped cavities cut in the walls of the shaft. The blocks of artificial stone are provided with inclined bearings *C*, which serve to transmit a portion of the downward thrust of the lining in the direction of the arrow.

Iron and Steel Supports.—The use of iron or steel, either for vertical or horizontal supports in mines, has not become at all general. In America, timber is as yet comparatively cheap in most mining localities, but this situation is fast changing and the timber reserves are being rapidly cut off, so that many mines now using wood must, in the comparatively near future, resort to some other form of support.

Trestles.—Figs. 31 and 32 illustrate the various timbers and methods of cutting the joints

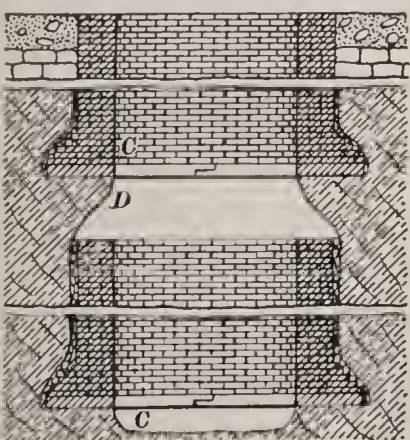


Fig. 29

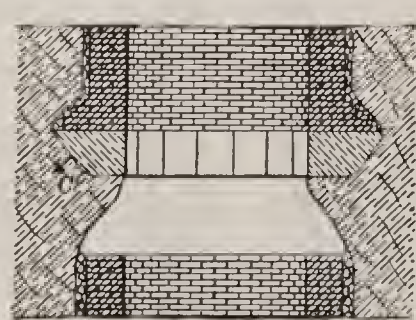


Fig. 30

for ordinary railroad trestles. In Fig. 32 (*a*) is shown the manner of framing a pile trestle, while (*b*) represents the manner of placing timbers and cutting the joint for the framed

trestle. Fig. 31 represents bents of a frame and pile trestle and the side elevation of a low pile trestle. Fig. 33 shows a bent

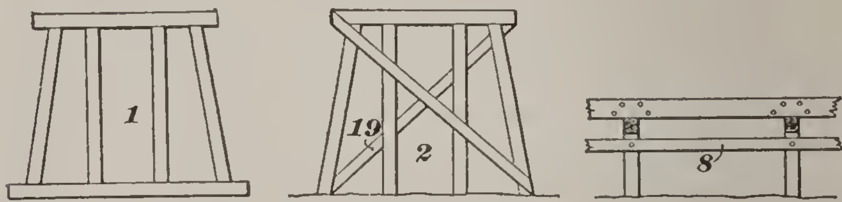


Fig. 31

of a framed trestle that is fastened together entirely by means of drift bolts, no joints whatever being cut.

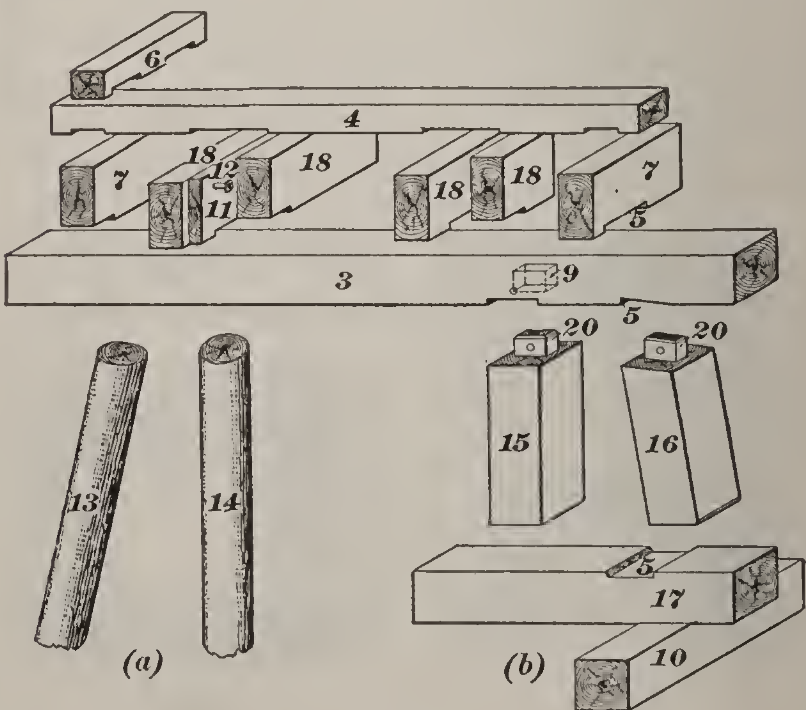


Fig. 32

The various parts of the trestles shown are numbered; their names are as follows:

1. Framed bent
2. Pile bent
3. Cap
4. Cross-tie
5. Gaining, dapping, or notching
6. Guard-rail

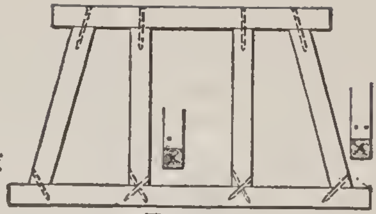


FIG. 33

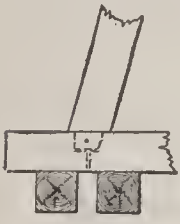


FIG. 34

7. Jack-stringer
8. Longitudinal brace or waling strip
9. Mortise
10. Mud-sill
11. Packing block
12. Packing bolts
13. Batter, inclined, or brace piles
14. Vertical, plumb, or upright piles
15. Vertical, plumb, or upright posts
16. Batter or inclined posts
17. Sill
18. Stringer
19. Sway-brace
20. Tenon

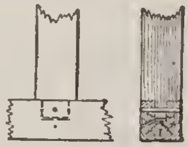


FIG. 35

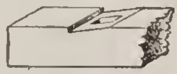


FIG. 36

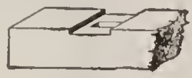


FIG. 37

Figs. 34 and 35 illustrate one manner of cutting the tenons and mortises on the ends of the batter braces and posts and frame bents, and also the drain holes that are bored in the mortise to prevent the timber from rotting. Usually the sills are

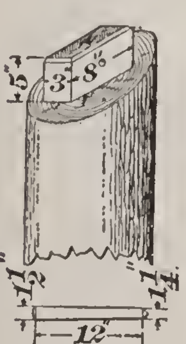


FIG. 38

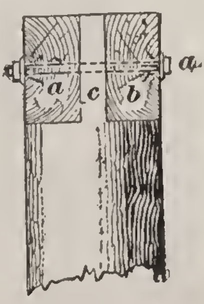


FIG. 39

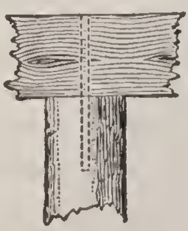


FIG. 40

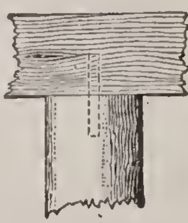


FIG. 41

notched or boxed to receive the ends of the timbers, in addition to having mortises formed in them. Figs. 36 and 37 show such

joints for receiving the batter brace and post. Fig. 38 shows how a tenon is sometimes formed on the top of the pile to secure the cap. When the cap is secured by a tenon, the two are united by a wooden pin shown in the lower part of the figure, known as a *treenail.*

Fig. 39 shows how the cap may be placed upon a pile trestle by splitting the cap into two pieces, *a* and *b* with the tenon *c* the full width of the pile between them. Fig. 40 shows how

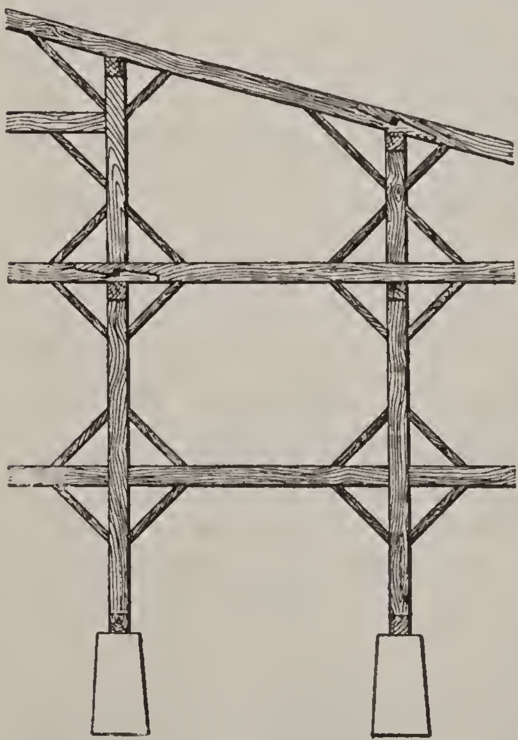


FIG. 42

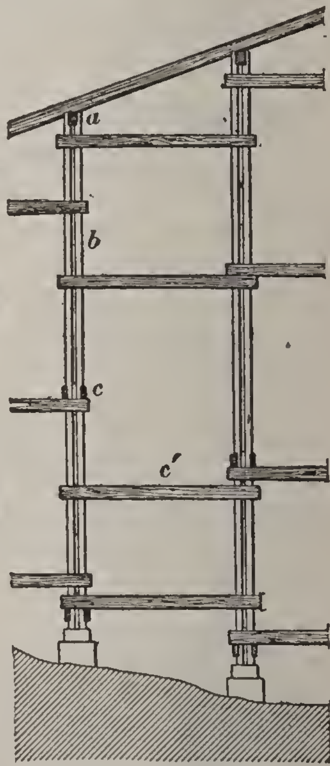


FIG. 43

the cap is sometimes secured to a pile by means of a drift bolt, and Fig. 41 shows how the same thing may be accomplished with the use of a dowel.

Figs. 42 and 43 show two methods of longitudinal bracing between the bents of the trestles for inclined planes, such as are used at breakers or concentrating mills. Fig. 44 is an elevation of a high trestle, showing the cross-bracing and framing of the structure.

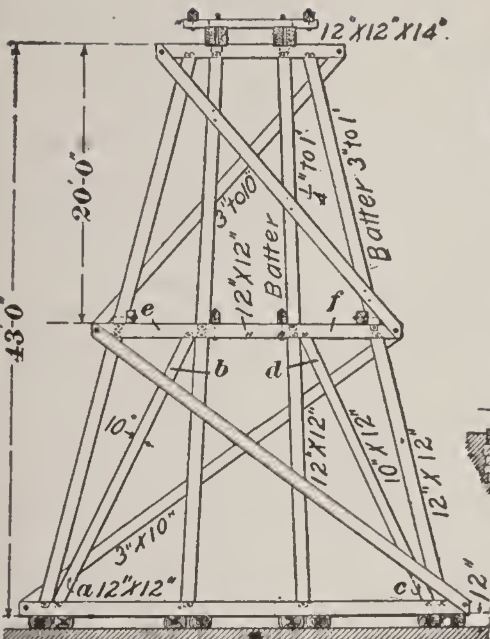


FIG. 44

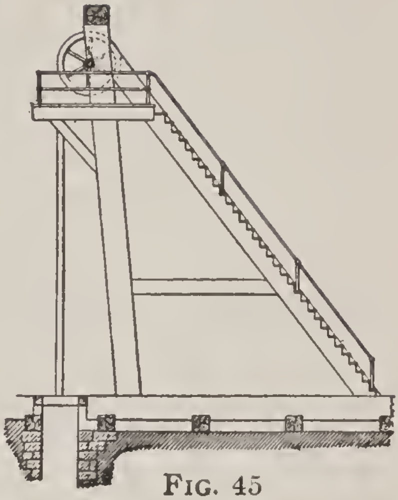


FIG. 45

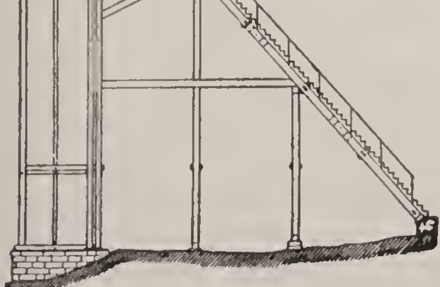


FIG. 46

FIG. 47

Timber Head-Frames or Head-Gears.—Fig. 45 is the simplest form of head-gear. This consists of a vertical post, which carries the weight of the sheave, etc., and a diagonal post that approximately bisects the angle between the rope from the drum and the rope hanging down the shaft, thus taking the resultant pull upon the axle of the sheave. There is usually some extra timbering, as shown, to support the cage guides and form a platform about the sheave for convenience in oiling.

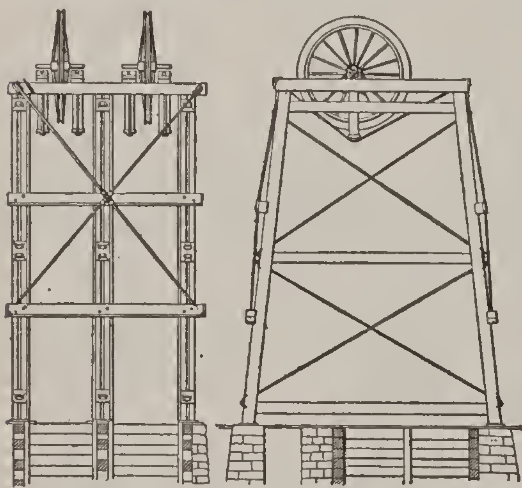


FIG. 48

Fig. 46 shows a modified form of the same type of frame, in which the main upright leg is vertical and in which there is also another vertical member on the opposite side of the shaft. The inclined leg is also braced and connected to the main vertical member. Fig. 47 is a head-frame for an inclined shaft where the coal or ore pocket is in the structure carrying the sheaves. Such head-frames are sometimes enclosed in their upper portions so as to protect the men during winter. Fig. 48 is a form of framing quite common in the anthracite fields of Pennsylvania, in which the timbers are further braced by tie-rods, as shown.

METHODS OF WORKING

GENERAL DESCRIPTION

No definite rules can be given for the selection of a method of mining that will cover all the conditions that may exist at any given mine. Each mine is a distinct and separate proposition, and each superintendent must adapt the general principles here given to the local conditions at his own mine. Every system of mining aims to extract the maximum amount of the deposit

in the best marketable shape and at a minimum cost and danger.

The elementary conditions affecting the extraction of coal are: (1) weight of overlying strata or depth of deposit, (2) strength and character of roof, (3) character of floor, (4) texture of bedded material, (5) inclination and thickness of bed, (6) presence of gas in seam or in adjoining strata.

Open Work.—Open work applies to the working of all deposits that have no overburden, or to those in which the overburden or overlying material is stripped from the portion of the deposit to be removed by hand, steam shovels, scrapers, etc. It includes particularly all quarries and placer workings, and can be applied to many mineral and coal deposits. Open work may be divided into two general classes: Where the whole or a greater part of the deposit is of value and has to be removed, as in quarries and in ordinary mines; where the valuable portion is but a small part of the whole, as in placers or fragmental deposits carrying gold, platinum, etc.

Closed Work.—Under the heading of closed work, it is customary to divide the methods of mining into *coal-mining methods* and *metal-mining methods*. This classification, however, is not entirely logical, for identical methods are applied regardless of the mineral. A more logical classification is one based on the position, character, and thickness of the deposit, but the older classification has become so firmly established that it is not advisable to disregard it entirely.

The typical and most extensive bedded mineral deposits are of coal and iron ore, and of these the former is by far the more extensively mined. A description of the several methods of mining coal beds will therefore comprise not only all the essential points in the mining of other bedded deposits, but will include a number of points not usually considered. The chief of these is the presence of explosive gas in such quantities as to influence the choice of a method of mining.

The *panel system* divides a mine into districts or panels by driving entries and cross-entries so as to intersect one another at regular intervals of, usually, about 100 yd. Large pillars are left surrounding the workings within each panel, and any method of development may be used for each panel. This

system has the following advantages: Better control of the ventilation, because the air in any panel may be temporarily increased or decreased, as required; an explosion occurring in one panel is less liable to affect the other workings. Coal may be extracted, pillars drawn, and the panels closed and sealed off independently of each other. Greater security is afforded against creep and squeeze. Coal that disintegrates on standing can be quickly worked out.

Bearing In, or Undercutting.—In any method of mining where the coal is undermined, advantage should be taken of the roof pressure to assist in breaking down the coal and in bearing in. The fact is often overlooked that the roof pressure upon the face coal makes it brittle and more susceptible to the pick, and the good miner starts a shallow mining in the under clay, or lower coal, and carries it the entire width of the face. Such a gradual system of mining throws the pressure on the coal face gradually, and the coal breaks in larger pieces. The depth of the undercut depends on the thickness of the seam and the other conditions.

SYSTEMS OF WORKING COAL

There are two general systems of working coal seams, the room-and-pillar, and the longwall. There are, however, a great number of modifications of each so that it is often difficult to exactly classify a given method.

Room-and-Pillar System.—The *room-and-pillar system*, also known as the *pillar-and-chamber* or *bord-and-pillar*, is the oldest system, and the one very generally used in the mines of the United States. The coal is first mined from a number of comparatively small places called *rooms*, *chambers*, *stalls*, *bords*, etc., which are driven either square from or at an angle to the haulageway. These openings may be wide or narrow, and may be a roadway, incline, or chute, according to existing conditions. The pillars that are left between the openings in the original workings support the roof, and usually are subsequently removed. All forms of room-and-pillar workings become impracticable when the thickness of the pillars necessary to support the roof pressure much exceeds double the width of the breast openings.

Fig. 1 shows a typical room-and-pillar method for working an approximately horizontal seam of coal of moderate thickness (4 to 10 ft.), and with a fairly good roof and bottom. The room openings are made suitable to prevailing conditions. The width of the room and the form of the opening depend on the character of the roof and the extent to which it is necessary to leave a pillar to support the cross-heading, it being advantageous, of course, to open out the room to its full width at the earliest possible moment.

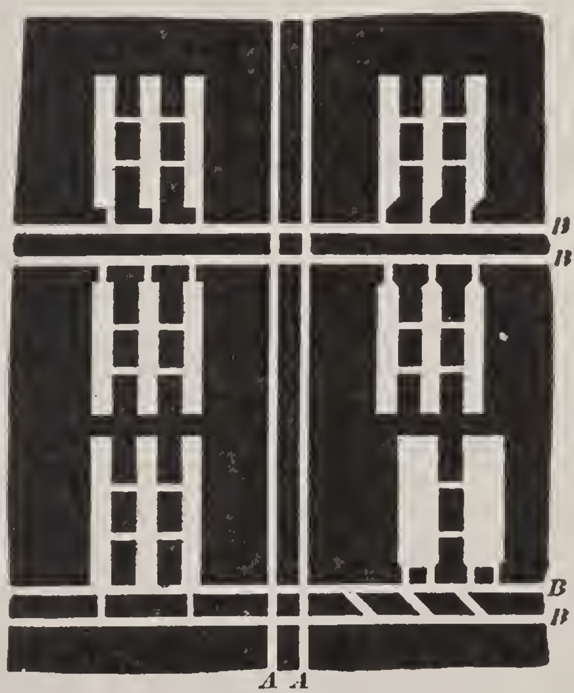


FIG. 1

The typical room-and-pillar plan, Fig. 1, shows the main headings and the rooms driven parallel to the direction of the dip, and the cross-headings parallel to the strike, but in most coal seams there are vertical cleavages, called *cleats*, which cross the coal in two directions about at right angles to each other. *Face cleats* are the more pronounced, while the *end*, or *butt*, *cleats* are the shorter, less pronounced joints. The direction of the face with respect to the cleats is of prime importance as greatly facilitating, or retarding the mining of the coal.

Fig. 2 shows the different positions that the face may occupy with respect to the direction of the cleats. The angle of the breast depends on the hardness of the coal and freedom of the cleats, and each method has its peculiar adaptation to the varying conditions of the coal seam. When the face cleats are working free and the coal is very soft, it may be necessary to drive end on. The *end-on method* is best adapted to a very heavy roof pressure, while for a light roof pressure the *short-horn* method assists in breaking the coal. If the face cleats are free and the coal breaks readily along them, and it is reasonably hard, the *long-horn* method is adopted, for when the coal is undercut it needs more support than it gets from the cleats, and its weight must be thrown somewhat upon the end cleats. *Face on* is adopted when the face cleats are not as free or numerous as the butt cleats.

FIG. 2

The *short-horn method* is adapted to heavy roof pressure and wide room pillars, as the face cleats are here quite pronounced, and the pillars between the rooms thereby weakened to a large extent; hence, wide pillars are more often employed when working on the ends of the coal.

Longwall Method of Mining.—In the *longwall system*, no portion of the seam is allowed to remain after leaving the vicinity of the shaft. The method depends on producing a uniform and gradual settlement of the roof a few yards behind the working face. In starting, the work of extraction may begin at the shaft itself, the coal being taken out all around and its place filled with solid packs, leaving only space for the roadways; or a pillar of solid coal cut only by the roadways, may be left to support the shaft. The longwall work may then be started uniformly all around this pillar.

Longwall may be advancing or retreating. In *longwall advancing*, mining begins at or near the foot of the shaft and

advances outwards, forming a gradually widening and increasing length of face to the boundary. The passages are made through the excavated portions of the mine, and are maintained by pack walls built either of the refuse secured in mining or from material brought in from the surface. Pack walls are built on each side of the roadways, and at regular intervals in the gob or waste area, and the roof settles firmly on these packs, pressing them into the bottom, or compressing them until the roof subsidence is complete. The height of the main roadway is maintained by *brushing* the roof or lifting the bottom. Longwall advancing is better suited to thin seams than to thick ones, to flat rather than pitching, and to good roofs and hard floors. In *longwall retreating*, entries, gangways, or headings are driven to the boundary or to other convenient distances inbye, and the pillars between these entries are then drawn back toward the shaft; this is also called *working home*. Longwall retreating is adapted to thick beds; to those liable to gob fires; to seams of hard coal having a considerable pitch; to pockety, or irregular seams; and to a soft and treacherous top. The air-course is also less broken along the face, and better haulage installations can be made. Its chief disadvantage is the large amount of dead work necessitated before returns are received. There is no expense in keeping up the haulage road so far as creep or falling roof is concerned, as the roads are all in solid coal, nor is there any trouble from gob fires or water; and little detriment to the working face is caused by the mine having to stand idle for a time. If the seam is high enough for the mules or horses, no rock whatever will need to be taken down. The coal seam will be proved before 10% of it is extracted. The ventilation in the retreating plan is as near perfect as it is possible to get it in practice. All the airways are tight, a thing impossible to get in the advancing plan; and it is a comparatively easy matter to shut off fire or to allow a portion of the working face to remain idle.

Longwall retreating is frequently used for working quite limited sections of a mine in which the seam of coal is 16 to 20 ft. thick, and, inclined not more than 10°. A series of 8 or 10 pairs of headings are turned off the butt entry and driven a distance, dependent on local conditions, where the working face

is formed by driving cross-cuts from one to the other. This face is carried back on the retreating plan, allowing the roof to cave in or settle on the gob as the work approaches the butt entry. In this way, any extra weight, that would crush and ruin the adjacent coal is avoided. This method is also used in

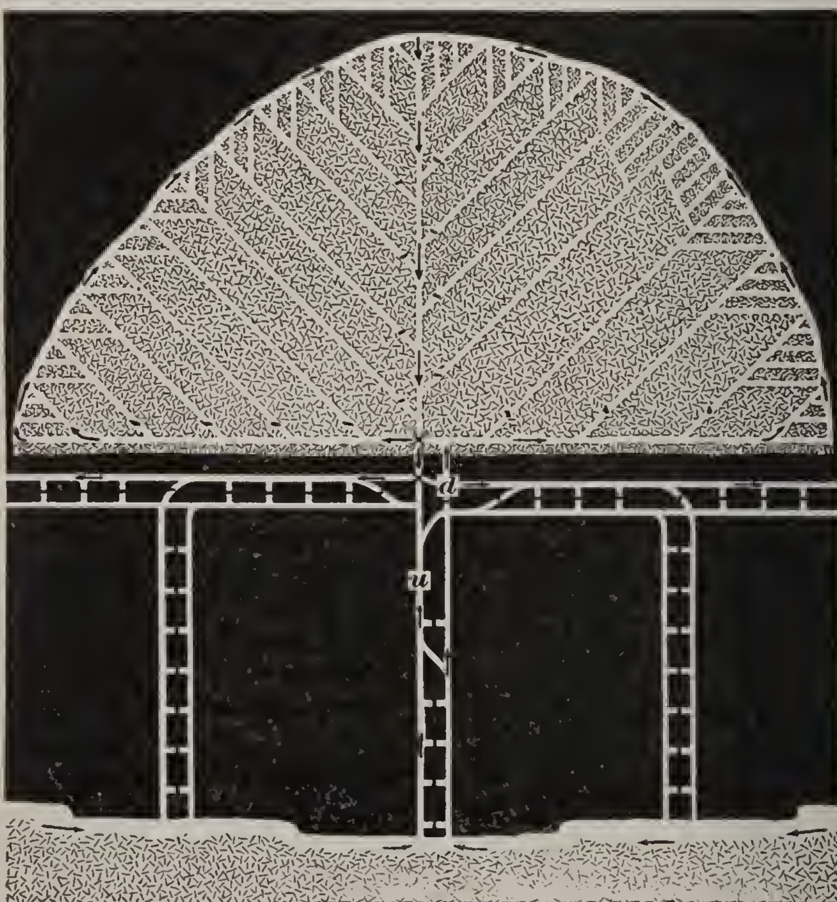


Fig. 3

lower seams in which the coal is soft, or the roof, or bottom, or both, are of such a nature as to give trouble in working the room-and-pillar system. Sometimes, instead of driving pairs of headings at considerable distances apart, a number of single headings are driven comparatively close together, and connected

by cross-cuts from 10 to 20 yd. apart. When the limit of the section is reached, the working face is formed and carried back, as in the other plan. This latter method is more suitable for tender roof, or a coal in which the face and butt cleats are not prominent.

Fig. 3 shows a plan of combined longwall advancing and retreating. In the upper arrangement, or Scotch longwall, the face is semicircular and the roads are turned off at angles of 45°. This plan is suitable for seams up to 3 ft. thick with a weak top, which pitch less than 20° and is situated at almost any depth. It is the one from which most of the longwall practice in the central coal basins of the United States is taken. In the lower arrangement headings from 200 to 300 ft. apart are driven in pairs to the boundary. Such a combination of longwall advancing and retreating insures an unvarying supply of coal, for while one side continually leaves the shaft, the other approaches it.

Timbering a Longwall Face.—The method of timbering the working face depends on the nature of the roof, floor, coal, etc. The action of the roof on the coal face is regulated almost entirely by timber; consequently, when the coal is of such a nature as to require little weight to make it mine easily, the roof must be timbered with rows of chocks and, if necessary, a few props.

Cribs, Pack Walls, and Stowings.—Pack walls should be built large enough at first and kept well up to the face, to prevent the weight coming upon the timber and also to permit the roof to settle rapidly when the timber is taken out of the face. Often the roof will not stand this second movement without breaking, and possibly closing in the entire face. The face should therefore be kept in shape, and just as soon as there is room for a prop or chock, it should be put in immediately, and the pack walls likewise should be extended after each cut or web is loaded out. No waste material, except such as will hasten spontaneous combustion, should be taken out of the mine to the surface.

Control of Roof Pressure.—The working face of a longwall working should advance up grade, but this face cannot always be kept parallel with the strike. When the angle at the line of

face, made with the line of strike, is less than 90°, the greater pressure of the covering rocks is thrown on the gob; when this angle is more than 90°, the greater pressure comes on the coal. The angle made by the working face with the line of pitch varies inversely as the vertical angle of pitch, or for a high pitch this angle is small and for a low pitch it is large. Where longwall is worked in adjacent sections, care must be taken to prevent the advancing of one section throwing a crushing weight on any of the others, and thus producing a crush or an uncontrollable cave. Where the rocks are pitching, and a greater portion of the cracks that cut them run in lines parallel to the strike, neither stone nor timber can efficiently support the roof, which frequently breaks off close to the working face.

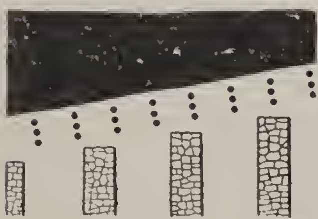


FIG. 4

The ends of all stone packs nearest the face of the coal should be in line, and the ends of these pack walls should form a line parallel to the face of the coal. Timbers set at equal distances and in line along a longwall face are much more efficient in supporting the roof than irregularly set timbers. Fig. 4 shows the proper way of locating the pack walls and the face timber.

NUMBER OF ENTRIES

The entries in a mine may be driven single, double, triple, etc. The single-entry system is only advisable under certain conditions and for short distances because the ventilation must be maintained along the face of the rooms, and there is but one haulage way, which may easily be closed by a fall or creep. Rooms are turned off on one or both sides of the entry. The *double-entry system* is most commonly used. Two parallel entries are driven, separated by an entry pillar whose thickness varies with the depth of the seam, and connected at intervals of about 20 yd. by cross-cuts or breakthroughs to maintain ventilation. The *triple-entry system* is used particularly in very gaseous seams requiring separate return airways; or, at times, in mines where the large output requires ample haulage roads. It is usually applied to the main entries only, but sometimes,

also, to the cross-entries. In gaseous mines, the middle entry is usually made the haulage road and intake airway, and the outside entries the return air-courses for either side of the mine, respectively. A still larger number of entries even has been suggested for deep workings where it is difficult to keep open broad passages, but these have not been generally adopted or tried experimentally to any great extent.

PILLARS

It is impossible to give exact rules or formulas for determining the proper size of pillars. Each case in practice requires special consideration, and in laying out the pillars in a virgin field it is well to find out what the current practice is in similar fields. In general, the thicker the seam and the greater its depth from the surface, the greater should be the thickness of the pillar.

Shaft Pillars.—Various formulas have been given to determine the size of shaft pillars, and the results given by these several formulas are very diverse.

Merivale's Formula.— $S=\sqrt{\frac{D}{50}}\times 22,$

in which S = length of side of pillar, in yards; D depth of shaft, in fathoms.

Andre's Formula.—Up to 150 yd. depth, have the pillar 35 yd. square, and for greater depths increase 5 yd. on each side for every 25 yd. of increased depth.

Dron's Formula.—Draw lines enclosing all surface buildings that it is necessary to erect about the head of the shaft, and make the shaft pillar so that solid coal will be left outside these lines all around for a distance equal to one-third the depth of the shaft.

Wardle's Formula.—Shaft pillars should not be less than 40 yd. square down to a depth of 60 fath. and should increase 10 yd. on a side for every 20 fath. increase in depth.

Hughes' Formula.—Leave 1 yd, in width of pillar for every yard in depth of shaft.

Pamely's Formula.—Allow a pillar 40 yd. square for any depth up to 100 yd.; for greater depths, increase the pillar 5 yd. for every 20 yd. in depth.

Calculating the size of pillar from each of these authorities gives the results shown in the accompanying table.

SIZE OF SHAFT PILLARS

Authority	For Shaft 300 Ft. Deep	For Shaft 600 Ft. Deep
Merivale	22 yd. square	31 yd. square
Andre	35 yd. square	45 yd. square
Wardle	40 yd. square	60 yd. square
Pamely	40 yd. square	65 yd. square
Dron	$33\frac{1}{3}$ yd. square*	$66\frac{2}{3}$ yd. square*
Hughes	100 yd. diameter	200 yd. diameter

*Outside of buildings.

None of these formulas takes account of the thickness of the seam, and the following formula, which takes account of this very important element, was suggested by Mr. R. J. Foster, in Mines and Minerals:

$$\text{Radius of pillar} = 3\sqrt{D \times t},$$

in which D = depth of shaft;

t = thickness of seam.

Pitching seams require smaller pillars on the low side than on the rising side of the shaft.

Room Pillars.—The relative width of pillar and breast is dependent on the weight of cover, as compared with the character of the roof and floor, and the crushing strength of the coal. These relative widths are determined largely by practice. Speaking generally, the narrower the rooms or chambers, the higher is the cost in yardage, the greater the production of slack and nut coal, the greater the consumption of powder, track iron, ties, etc., and the greater the cost of dead work.

For bituminous coal of medium hardness and good roof and floor, a rule often used is to make the thickness of room pillars equal to 1% of the depth of cover for each foot of thickness of the seam, according to the expression

$$W_p = \frac{t}{100} \times D,$$

in which Wp = pillar width;

t = thickness of seam;

D = depth of cover.

Then the width of breast or opening is made equal to the depth of cover divided by the width of pillar thus found, according to the expression

$$W_o = \frac{D}{W_p},$$

in which W_o = width of room.

The accompanying table is for first working, with the design of afterwards taking out the pillars, the width of the principal workings being 5 yd., and cross-holings 2 yd.

DUNNS' TABLES OF SIZE OF ROOM PILLARS FOR VARIOUS DEPTHS

Depth Feet	Size of Pillars Yards	Proportion in Pillars	Depth Feet	Size of Pillars Yards	Proportion in Pillars
120	20×5	.41	1,080	26×14	.69
240	20×6	.50	1,200	26×16	.71
360	22×7	.52	1,320	28×18	.73
480	22×8	.57	1,440	28×20	.75
600	22×9	.59	1,560	30×21	.77
720	22×12	.61	1,680	30×22½	.78
840	26×15	.63	1,800	30×24	.79
960	28×16	.66			

Extremely large pillars must often be left as a precautionary measure to protect permanent haulage ways and surface buildings, or to avoid any possibility of a break in the roof that would cause an inflow of water.

Drawing Pillars.—Drawing pillars is about the most dangerous work the miner has to perform, but the fact of its being so is no doubt the reason why, comparatively speaking, so few serious accidents happen in it. It is not so much that the best, most skilled workmen are chosen to perform pillar drawing, as that the men, being alive to the dangers, are more on the alert and careful to protect themselves. Methods of drawing

DISTANCE BETWEEN CENTERS OF BREASTS OR CHAMBERS MEASURED ON ENTRY OR GANGWAY, FOR DIFFERENT ANGLES

Angle Between Chamber and Entry Degrees	Width of Breast + Width of Chamber											
	20	25	30	35	40	45	50	55	60	65	70	75
	Distance Measured on Entry, in Feet											
90	20.0	25.0	30.0	35.0	40.0	45.0	50.0	55.0	60.0	65.0	70.0	75.0
85	20.0	25.1	30.1	35.1	40.2	45.2	50.1	55.2	60.2	65.3	70.3	75.3
80	20.3	25.4	30.5	35.5	40.6	45.7	50.6	55.8	60.9	66.0	71.1	76.2
75	20.7	25.9	31.1	36.2	41.4	46.6	51.2	56.9	62.1	67.3	72.5	77.7
70	21.2	26.6	31.9	37.2	42.6	47.9	53.1	58.5	63.9	69.2	74.5	79.8
65	22.0	27.6	33.1	38.6	44.1	49.6	55.1	60.7	66.2	71.7	77.2	82.8
60	23.0	28.9	34.6	40.4	46.2	52.0	57.6	63.5	69.3	75.1	80.8	86.6
55	24.4	30.5	36.6	42.7	48.8	54.9	60.9	67.1	73.3	79.4	85.5	91.6
50	25.8	32.6	39.2	45.7	52.2	58.7	65.1	71.8	78.3	84.9	91.4	97.9
45	28.2	35.4	42.4	49.5	56.6	63.6	70.6	77.8	84.9	91.9	99.0	106.1
40	31.1	38.9	46.7	54.5	62.2	70.0	77.6	85.6	93.4	101.2	109.0	116.7
35	34.9	43.6	52.3	61.0	69.7	78.5	87.0	95.9	104.6	113.4	122.1	130.8
30	40.0	50.0	60.0	70.0	80.0	90.0	100.0	110.0	120.0	130.0	140.0	150.0
25	47.3	59.2	71.0	82.8	94.6	106.5	118.1	127.2	142.0	153.8	165.7	177.5
20	58.5	73.1	87.7	102.4	117.0	131.6	145.9	160.8	175.5	190.1	204.7	219.3
15	77.4	96.6	115.9	135.3	154.5	173.9	192.8	212.5	231.9	251.2	270.5	289.8
10	115.2	144.0	172.8	201.6	230.4	259.2	287.3	316.7	345.6	374.4	403.1	432.0
5	229.5	286.9	344.2	401.6	459.0	516.3	572.5	631.1	688.5	745.8	803.2	860.5

pillars vary according to the inclinations of the seams, the nature of the roof and floor, and the character of the coal; Figs. 1 and 2 show the common methods. In Fig. 1, at *A*, *B*, and *C*, the drawing begins by cross-cutting the fast ends

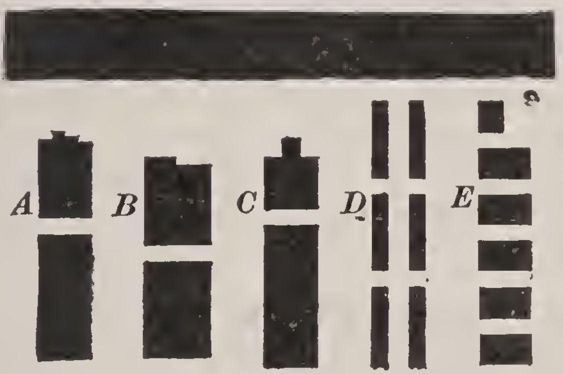


FIG. 1

of the pillars to obtain a retreating face. At *A* is shown a method for soft coal and narrowing pillars; at *B*, a method for wide pillars, the end being taken in two lifts; while the method at *C* is for harder coal and shows it taken in three lifts. At *D* and *E* the pillars are cut into stocks to be drawn by side or end lifts, according to the character of the coal, the inclination

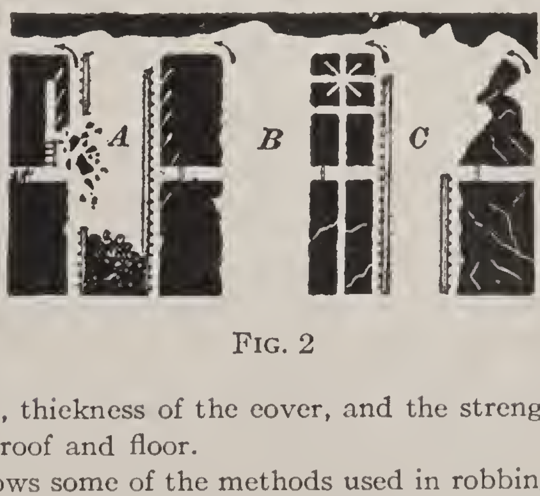


FIG. 2

of the seam, thickness of the cover, and the strength or weakness of the roof and floor.

Fig. 2 shows some of the methods used in robbing the pillars in steep pitching, thick beds of anthracite. To get the coal

out of the pillar at the left of A, a skip is taken off the side, as shown. Successive skips are thus taken off until the whole is removed, the miner keeping the manway open to the heading below as a means of retreat. The pillar between A and B is similarly worked. To remove that between B and C, a narrow chute or heading is driven up the middle, and cross-cuts put to the right and left a few yards from the upper end. Shots are placed in the four blocks of coal thus formed, as shown, and they are fired simultaneously by battery. This operation is repeated in each descending portion unless the pillar begins to run. A pillar from which the coal has started to run is shown to the right of C.

SPONTANEOUS COMBUSTION

The cause of the *spontaneous ignition* of coal, chiefly, is the condensation and absorption of oxygen from the air by the coal, which of itself causes heating, and this promotes the chemical combination of the volatile hydrocarbons in the coal and some of the carbon itself with the condensed oxygen. Another cause is moisture acting on sulphur in the form of iron pyrites. The heating effect of this cause is very small, and it acts rather by breaking the coal and presenting fresh surfaces for the absorption of oxygen.

Gob fires are due to the spontaneous ignition of coal, and are most likely to occur in pack walls and gobs where there is an insufficiency of air. Ample ventilation is the best preventive.

COAL STORAGE

The coal store should be well roofed in, and have an iron floor bedded in cement. All supports passing through and in contact with the coal should be of iron or brick; if hollow iron supports are used, they should be cast solid with cement. The coal must never be loaded or stored during wet weather, and the depth of coal in the store should not exceed 8 ft.. and should only be 6 ft. where possible. Under no condition must a steam or exhaust pipe or flue be allowed in or near any wall of the store, nor must the store be within 20 ft. of any boiler, furnace, or bench of retorts. No coal should be stored or shipped to distant ports until at least 1 mo. has elapsed since

it was brought to the surface. Every care should be taken during loading or storing to prevent breaking or crushing of the coal, and on no account must a large accumulation of small coal be allowed. These precautions, if properly carried out, would amply suffice to entirely do away with spontaneous ignition in stored coal on land.

When the coal pile has ignited, the best way to extinguish the fire is to remove the coal, spread it out, and then use water on the burned part. The incandescent portion is invariably in the interior, and when the fire has gained any headway usually forms a crust that effectually prevents the water from acting efficiently.

MODIFICATIONS OF ROOM-AND-PILLAR METHODS

Some modifications of the room-and-pillar plan shown in Fig. 1 can usually be applied to seams whose dip does not exceed 3°. When the pitch is greater, rooms are often turned off toward the rise only, and the cross-entries driven correspondingly closer together. When the pitch is from 5° to 10°, the cars may still be taken to the face if the rooms are driven across the pitch, thus making an oblique angle with an entry or gangway, the rooms being known as *room breasts*.

Buggy Breasts.—For inclinations between 10° and 18°, that is, after mule haulage becomes impossible and until the coal will slide in chutes, buggies are often used. Fig. 1 shows a *buggy breast*, in plan and section. Coal is loaded into a small car or buggy *c*, which runs to the lower end of the breast and there delivers the coal upon a platform *l*, from which it is loaded into the mine car. The refuse from the seam is used in building up the track, and if there is not sufficient refuse for this, a timber trestle is used.

Another form of buggy breast is shown in Fig. 2. Here the coal is dumped directly into the mine car from the buggy. If the breast pitches less than 6°, the buggy can be pushed to the face by hand, but in rooms of a greater pitch, a windlass is permanently fastened to timbers at the bottom of the breast, while the pulleys at the face are temporarily attached to the props by chains, so that they can be advanced as the face advances. The rope used is from $\frac{1}{2}$ to $\frac{5}{8}$ in. in diameter, and

any form of ordinary horizontal windlass can be used. With the windlass properly geared, one man can easily haul a buggy to the face of a breast in a few minutes. The buggy runs upon 20-lb. T rails spiked with $2\frac{1}{2}$ in. $\times \frac{3}{8}$ in. spikes upon 2 in. $\times$ 4 in. hemlock studding sawed into lengths of 14 ft.

Chute Breasts.—Seams pitching more than 15° are usually worked by chutes, or self-acting inclines. When the pitch is between 15° and 30°, sheet iron is laid to furnish a good slid-

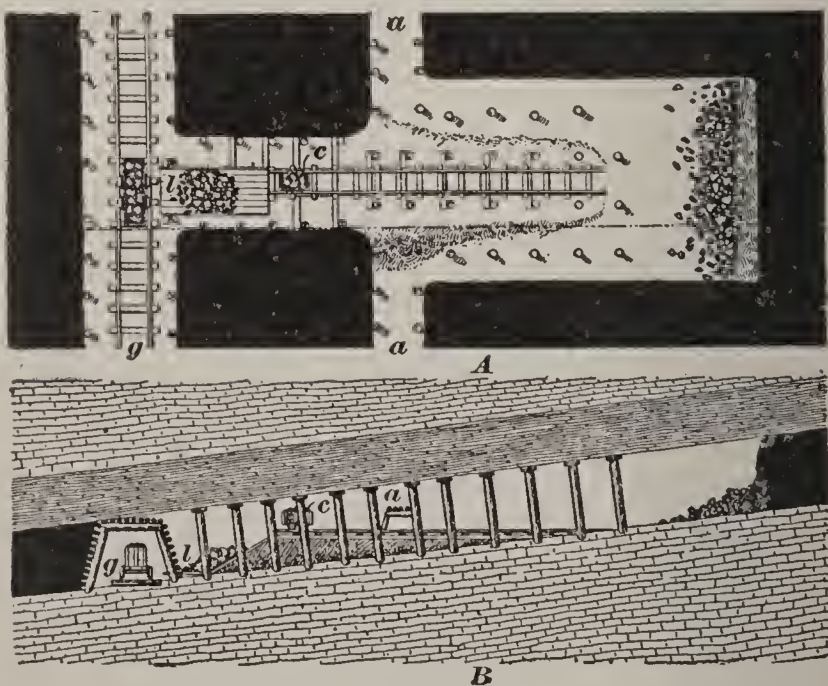


FIG. 1

ing surface for the coal. On inclinations of less than 18° to 20°, it is usually necessary to push the coal down the chute. Sheet iron is not required on pitches above 30°. It must be remembered that these pitches are only fair averages, as much depends on the character of the coal. Anthracite slides more easily than bituminous.

To secure the best returns from a coal seam, the slope or shaft should be driven to the basin, and the lowest gangways or levels first driven to the property limits, and the coal then

worked retreating toward the slope or shaft. Practice is, however, usually contrary to this, and the upper levels or gangways are turned off first, and working places opened out as rapidly as the gangway is driven. Fig. 3 shows a method of grouping rooms that may be used where the pitch is from 8° to 20°, the straight heading being driven on the strike

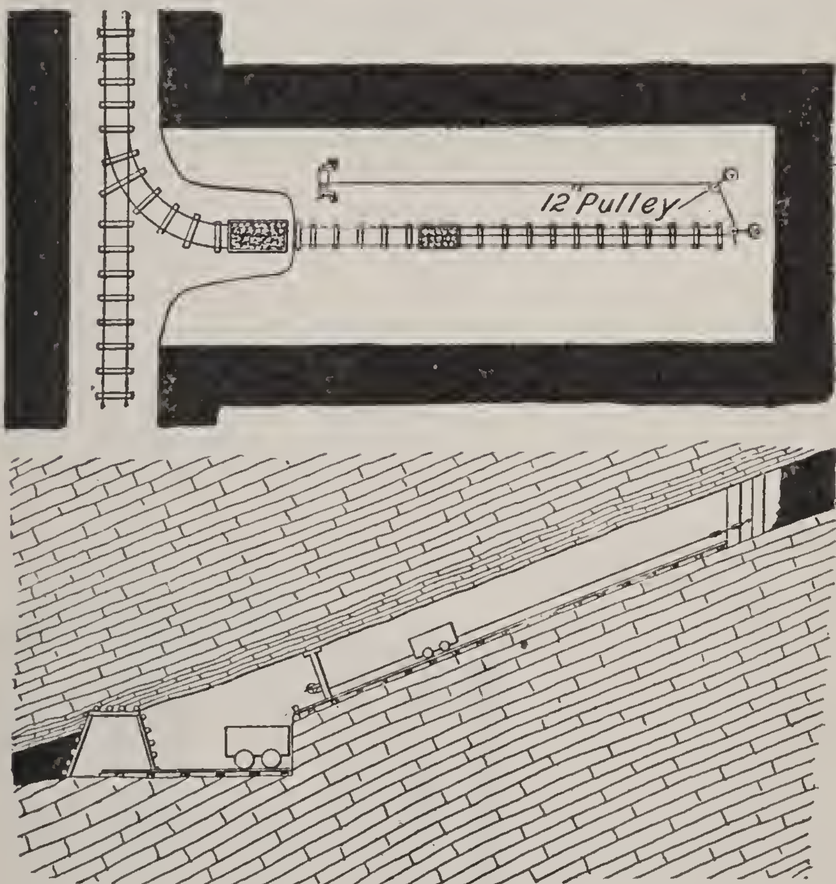


FIG. 2

and the other headings at such angles as will give a good grade for haulage purposes.

Pillar-and-Stall Method.—The pillar-and-stall system is a modification of the room-and-pillar, to which it is similar in all respects excepting in the relative size of the pillars and breasts. The stalls are usually opened narrow and widened

inside, according to conditions of roof, floor, coal, depth, etc., being from 4 to 6 yd. in the single-stall method, with the pillars about the same width. Fig. 4, *A* and *B*, shows single and double stalls. This system is adapted to weak roof and floor, or strong roof and soft bottom, to a fragile coal, or wherever ample support is required, and is particularly useful in deep seams with great roof pressure. Double stalls are often driven from 12 to 15 yd. wide, with an intervening pillar of sometimes 30 yd. Following are a few applications of the pillar-and-room method carried out in some of the leading coal fields of America:

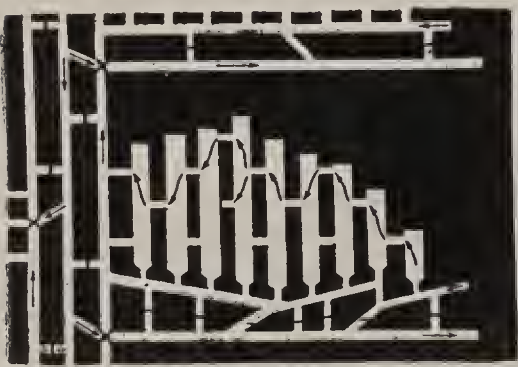


FIG. 3

Connellsville Region.—Fig. 5 shows the common method used in Connellsville, Pa., region, where the average dip is about 5%. The face and butt headings are driven, respectively, at right angles to each other on the face and the butt of the coal. The face headings leave the main butts about 1,000 ft. apart, while from these face headings, and 400 ft. apart, secondary butts are driven, and again from these butts on the face of the coal the rooms or wide workings are excavated to a length of 300 ft., this having proved the most convenient length for economical working. Room pillars have a thickness of 30 to 40 ft., while the rooms

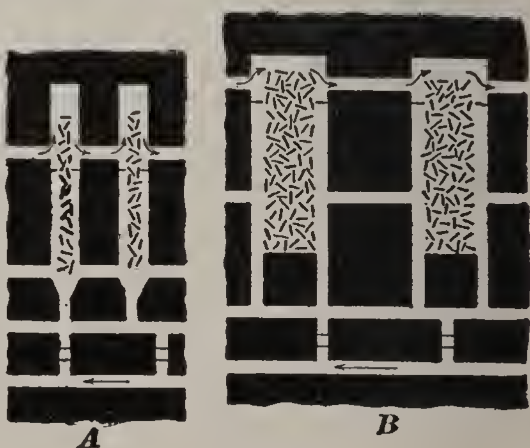


FIG. 4

are 12 ft. in width and are spaced 42 to 52 ft. between centers, depending on depth of strata over the coal. The headings are 8 ft. wide, and in all main butts and faces the distance between centers of parallel headings is 60 ft., leaving a solid rib of 52 ft.

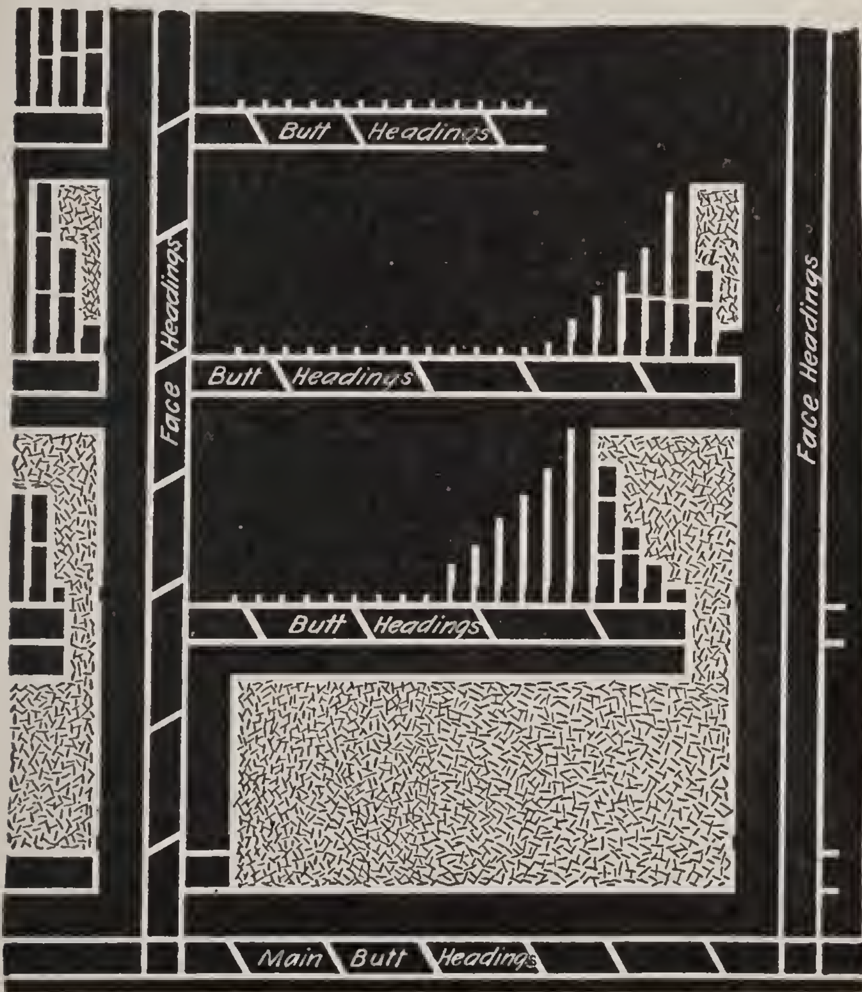


FIG. 5

A solid rib of 60 ft. is also left on the side of each main heading.

The method of drawing ribs is one of the advantages of the system, as it is harder to do successfully in a soft coal like the Connellsville coal than in hard coal. The coal itself is firm.

When necessary to protect the top or bottom, 4 to 6 in. of coal are left covering the soft material.

The method just given is often applied to a whole series of butts (4 or 5) at once instead of to each butt in turn. In this case, work is started at the upper end of the uppermost butt and progresses, as shown; but, after cutting across the butt heading from which the rooms were driven, the butt heading itself and the upper rooms from the second butt, or that just before, are drawn back by removing continuous slices from the rooms of the upper butt, and on across the next lower butt, etc., all on an angle to the butts, until another butt is reached, etc. This gradually makes a longer line of fracture, which is only limited by the number of butts it is desired to include at one time in the section thus mined.

Pittsburg Region.—The coal is worked in much the same way as in the Connellsville region, except that a different system of drawing ribs is used. The coal is worked on the room-and-pillar system, with double entries, with cut-troughs between for air, and on face and butt. Entries are about 9 ft. wide, and the rooms 21 ft. wide and about 250 ft. long; narrow (or neck) part of room, 21 ft. long by 9 ft. wide. Room pillars are 15 to 20 ft. wide, depending on depth of strata over the coal, which is from a few feet to several hundred feet. The mining is done largely by machines of various types. Coal is hard, of course, and, in many places, the roof immediately over the coal is also quite hard. There are about 4 ft. of alternate layers of hard slate and coal above the coal seam. Rooms are mined from lower end of butt as fast as butt is driven, the ribs being drawn as mining progresses. As the coal is harder than in the Connellsville region, thickness of coal pillar between parallel entries is somewhat less.

Clearfield Region.—The butt and face are not strongly marked in the B or Miller seam, the one chiefly worked in the Clearfield region. Where possible, these cleavages are followed in laying out the workings, but the rule is to drive to the greatest rise or dip and run headings at right angles to the right and left, regardless of anything else. The main dip or rise heading is usually driven straight, and is raised out of swamps or cut down through rolls—very common here—unless they

are too pronounced, when the heading is curved around them. The same is true of room headings, except that they are more usually crooked, not being graded except over very minor disturbances.

Reynoldsville Region.—The average thickness of the principal seam is $6\frac{1}{2}$ ft. and the pitch is 3° to 4°. The coal is hard and firm, and contains no gas; the cover is light, and on the top of the coal there are 3 or 4 ft. of bony coal; the bottom is fireclay. Drift openings and the double-entry system are used. Both main and cross entries are 10 ft. wide, with a 24-ft. pillar between. The cross-entries are 600 ft. apart, and a 24 ft. chain pillar is left along the main headings. The rooms are about 24 ft. wide and open inbye, the necks being 9 ft. wide and 18 ft. long. The pillars are from 18 to 30 ft. thick.

West Virginia.—In the northern part of West Virginia, the coal measures vary from 7 to 8 ft. in thickness, and have a covering varying from 50 to 500 ft. The coal does not dip at any place over 5%. In most places the coal is practically level, or has just sufficient dip to afford drainage. The usual method of exploitation is to advance two parallel headings, 30 ft. apart, on the face of the coal. At intervals of 500 to 600 ft., cross-headings are turned to right and left, and from these headings rooms are turned off. These cross-headings are driven in pairs about 20 or 30 ft. apart. Between the main headings and the first room is left a block of coal about 100 ft., and on the cross-headings there is often left a barrier pillar of 100 ft. after every tenth room.

The headings are driven from 8 to 12 ft. wide, and the rooms are made 24 ft. wide and 250 to 300 ft. long. A pillar is left between the rooms about 15 to 20 ft. wide. These pillars are withdrawn as soon as the panel of rooms has been finished. The rooms are driven in from the entry about 10 ft. wide for a distance of 20 ft., and then the width is increased on one side. The track usually follows near the rib of the room. Cross-cuts on the main and cross headings are made every 75 to 100 ft., and in rooms about every 100 ft. for ventilation.

The double heading system of mining and ventilation is in vogue. Overcasts are largely used, but a great many doors are

used in some of the mines. Rooms are worked in both directions, when the grades are slight, but when the coal dips over 1%, the rooms are driven in one direction only; in this case, they are made as much as 350 ft. long. It is the custom then to break about every third room into the cross-heading above. The floor of this bed of coal, being composed of shale and fireclay, often heaves, especially when it is made wet. Some

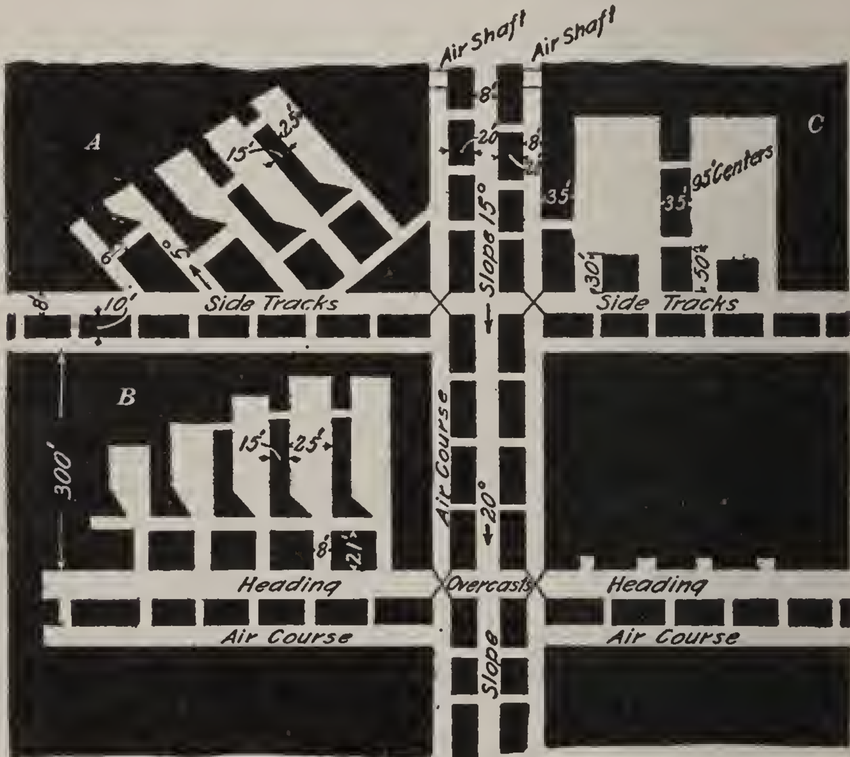


FIG. 6

trouble is at times experienced by having the floor heave by reason of the pillars being too small for the weight they support.

Alabama Methods.—Fig. 6 shows the common methods used in working the Alabama coals. The seams now working vary from 2 to 6 ft. thick, and they pitch from 2° to 40°. Where the seams are thin, the coal is hard, and pillars of about 20 to 30 ft. are used to support the roof. The rooms are worked across the pitch on an angle of about 5° on the rail, Fig. 6, *A*, when the

coal does not pitch greater than 20°; where the pitch is greater, chutes are worked and the rooms are driven straight up the pitch, as in Fig. 6, *B*. In a few cases, where the pitch is not greater than 15°, double rooms are worked with two roadways in each room, as in Fig. 6, *C*.

George's Creek District, Md.—Fig. 7 shows the method used in the George's Creek field, Md. The coal shows no indication of cleats, and the butts and headings can be driven in any direction. The main heading is driven to secure a light grade for hauling toward the mouth. Cross-headings making an angle of 35° to 40° are usually driven directly to the rise, and of dimensions shown.

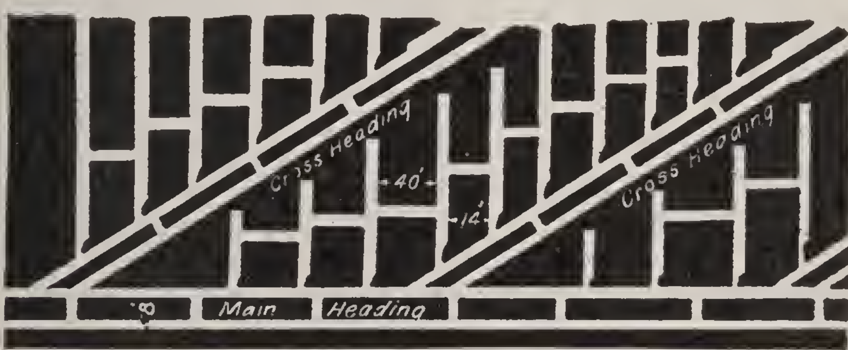


FIG. 7

Indiana Coal Mining.—Fig. 8 shows the method as used in Indiana. The entries are generally 6 ft. high, 8 ft. broad, the minimum height required by law being 4 ft. 6 in. The rooms are from 21 to 40 ft. in width. The mines are generally shallow. The rooms in Fig. 8 are shown as widened on both ribs, but a more usual method in this locality is to widen the room on the inbye rib, leaving one straight rib for the protection of the road in the room.

Iowa Coal Mining.—The entry pillars along the main roads are 6 to 8 yd. thick, for the cross entries 5 to 6 yd., and for the rooms 3 to 5 yd. Room pillars are drawn in when approaching a cross-cut. Both room-and-pillar and longwall methods are in use, with modifications of each. In the room-and-pillar method, the double-entry system is almost invariably used in

the larger mines. Rooms are driven off each entry of each pair of cross-entries at distances of 30 to 40 ft., center to center. The rooms are 8 to 10 yd. in width, and pillars 3 to 4 yd. The rooms are narrow for a distance of 3 yd., and then widened inbye at an angle of 45° to their full width. They vary from 50 to 100 yd. in length, and the road is carried along the straight rib.

When double rooms are driven, the mouths of the rooms are 40 to 50 ft. apart, and they are driven narrow from the entry a distance of 4 or 5 yd. A cross-cut is then made connecting them, and a breast 16 yd. wide is driven up 50 to 60 yd. The

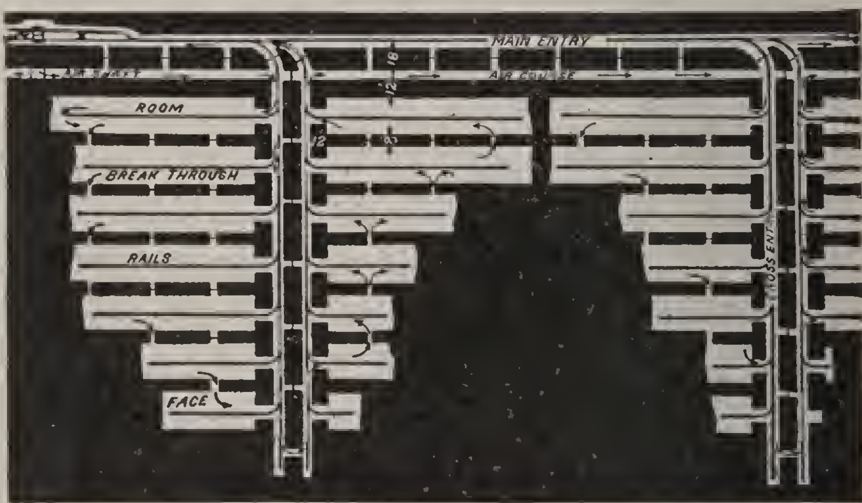


Fig. 8

pillar between each pair of rooms is 12 to 15 yd. In some cases, the stalls are usually turned off narrow and widened inside, the pillar varying from 5 to 8 yd. The stalls are 30 to 40 yd. in length, and the pillars are drawn back. When the stalls are driven in pairs, the pillar 8 to 10 yd. in width is carried between them.

When the longwall system is used, the main haulage road runs in each direction from the foot of the shaft, and on both sides of this diagonal roads are turned at an angle of 45°, or parallel to the main haulageway. These are spaced 10 yd. apart and driven 50 to 60 yd., when they are cut off by another

diagonal road. Panel breasts are used where the conditions are such as to induce a squeeze. Rooms are turned narrow off entries and are arranged in sets of 6 to 12 rooms, with a pillar 10 to 20 yd. wide between the sets of rooms. When the rooms have progressed a short distance from the entry, they are connected by cross-cuts, and the longwall face is carried forwards from this point. Packs are built and the roof allowed to settle, as in longwall. The wide pillars are taken out after the roof has settled.

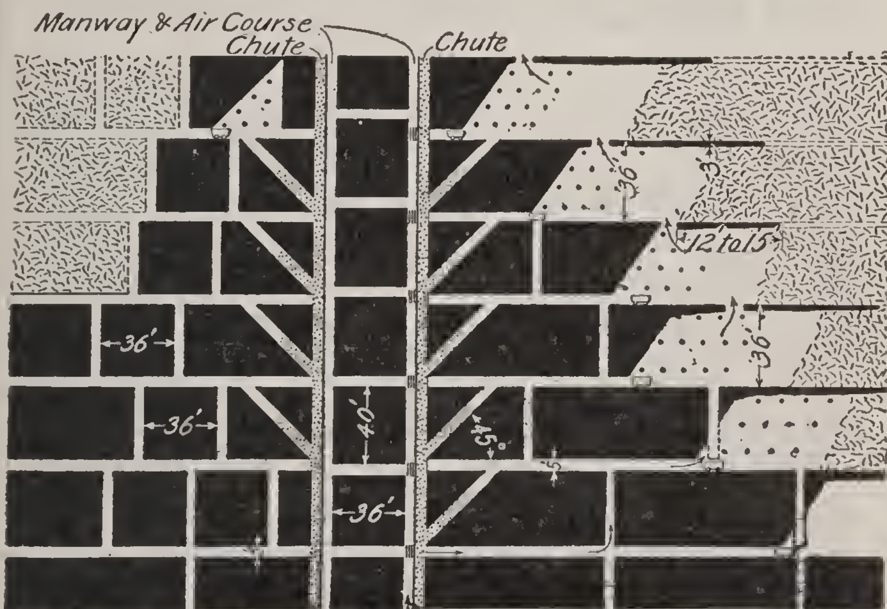


FIG. 9

Tesla, Cal.—The Tesla, Cal., method is shown in Fig. 9. The coal seam averages 7 ft. of clear coal, and pitches 60°. This system was adopted in a portion of the mine to get coal rapidly; for, at this point, a short-grained, slate cap rock came in over the coal, making it difficult to keep props in place. The floor is a close blue slate and has a decided heaving tendency. The roof is an excellent sandstone. There is a small but troublesome amount of gas. Two double chutes are driven up the pitch at a distance of 36 ft. apart, connected every 40 ft. by

cross-cuts. One side of each chute is used for a coal chute and the other for a manway and air-course. At a distance of 12 yd. apart small gangways are driven parallel with the main mine gangways. These are continued from each chute a distance of 300 ft., if the conditions warrant it. The top line is then attacked from the back end and the coal is worked on the cleavage planes; the breast, or room, therefore consists of a

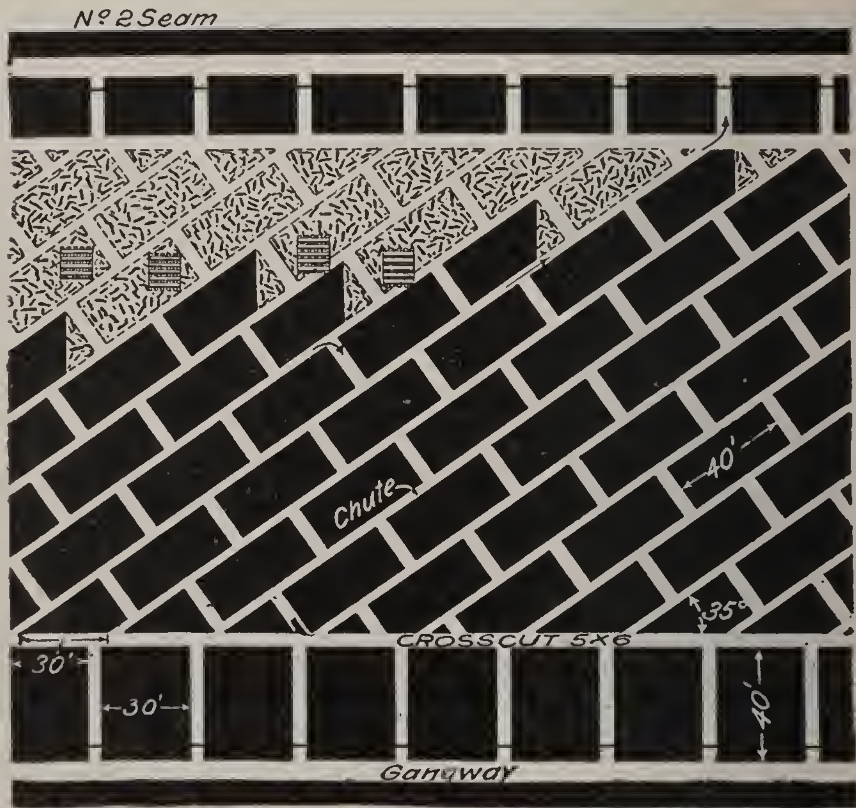


FIG. 10

12-yd. face, including the drift or gangway through which the coal is carried to the chutes; a rib of coal (2 or 3 ft.) is left between the breasts to keep the rock from falling on the breast below. Thus in each breast the miners have a working face of about 15 or 16 yd., and as the coal is directed to the car by a light chute, moved along as the face advances, the coal is delivered into the cars at small cost, and but little loss results

from the falling coal, as a minimum of handling is thus obtained. Fig. 10 shows another system used in No. 7 vein at the same place. The seam averages 7 ft. of coal. The roof is shelly and breaks quickly, hence the coal must be mined rapidly.

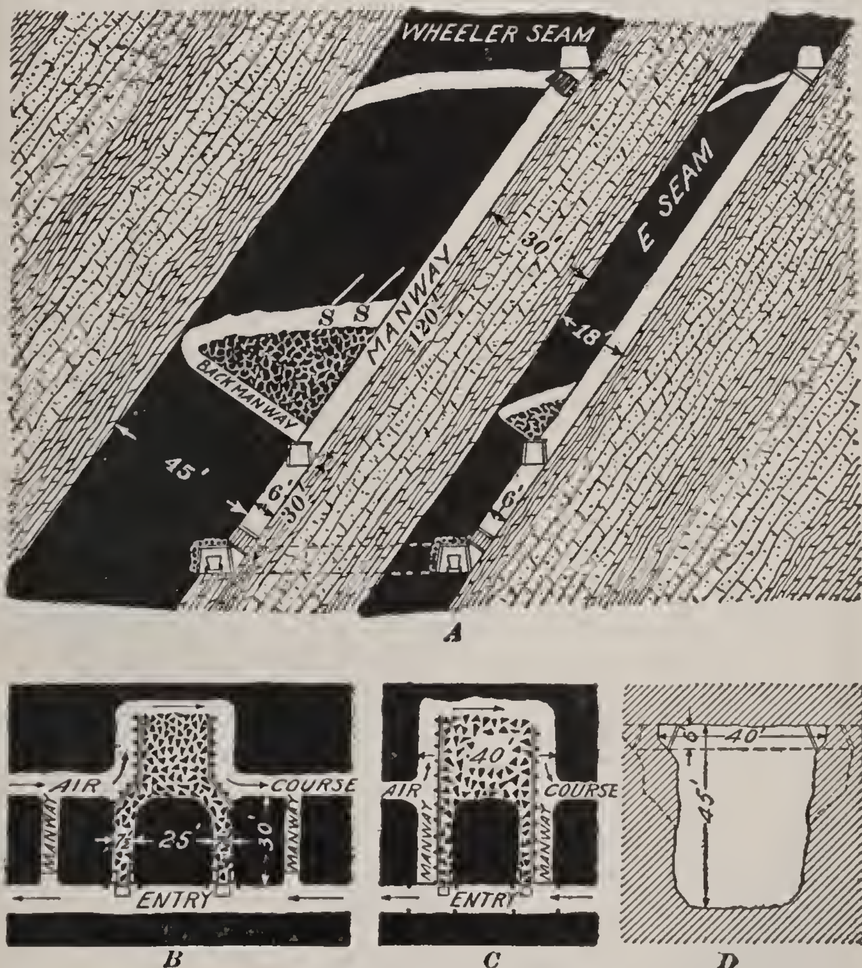


FIG. 11

New Castle, Colo.—The following method is used at New Castle, Colo., for highly inclined bituminous seams. The coals mined are only fairly hard, contain considerable gas, and make much waste in mining. Fig. 11 shows the method used for extracting the Wheeler or thicker vein to its full width of 45 ft.,

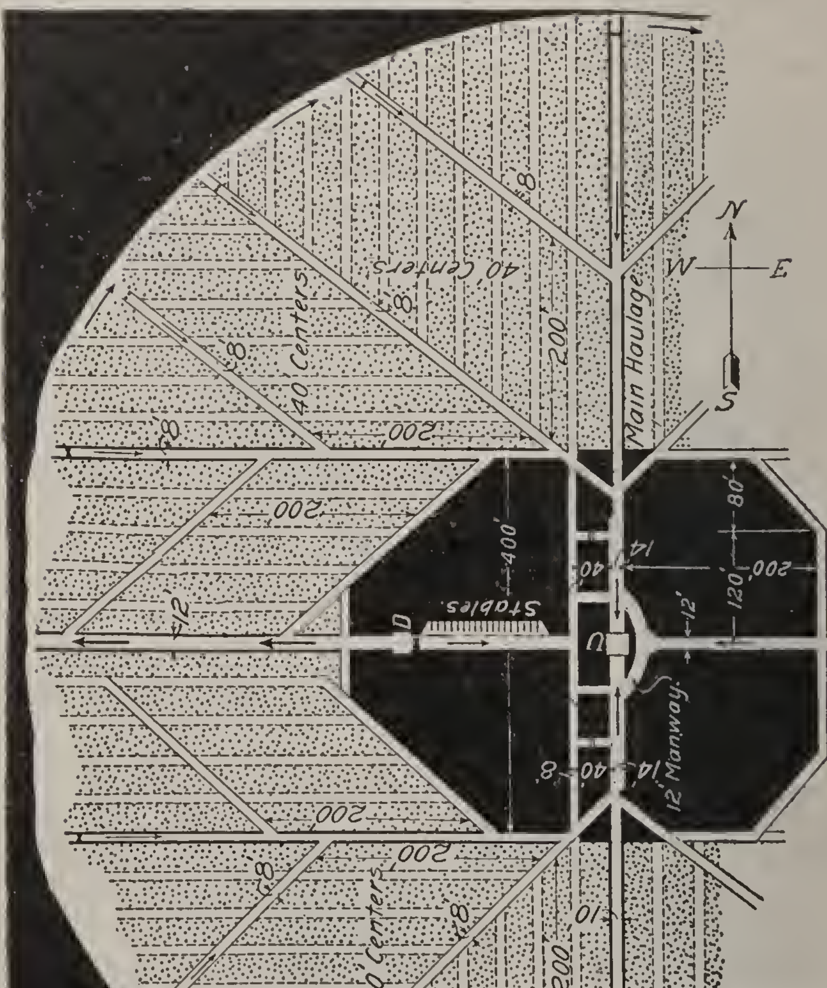

Main Haulage
Main Haulage
40' Centers
40' Centers
40' Centers
40' Centers
Stables
12' Manway
400'
200'
120'
80'
12'
N
W
E
S
D
U

FIG. 12

and the E seam 18 ft. thick, excepting that left for pillars. Rooms and pillars are laid out under each other in the two seams whenever practicable. Entries are along the foot-wall; 30 ft. up the pitch is an air-course. Rooms and breasts are laid out as shown in *B* and *C*.

MODIFICATIONS OF LONGWALL METHOD

Fig. 12 shows a good arrangement of the main and temporary haulageways in a flat seam. The chief object in any plan of longwall workings is to have the permanent roadways the arteries of the system, providing the most direct route from all sections of the mine to the shaft. The temporary roads or working places are only maintained for a distance of 60 to 100 yd., until cut off by subroads branching at regular intervals from the main roads. The full heavy lines indicate the permanent haulageways, except only the main intake airway (12 ft. wide), running west from the downcast shaft *D*, and the main return air-course (12 ft. wide) leading from the face on the east side to the manway around the upcast *U*, which is the hoisting shaft. The full light lines indicate the diagonal subroads, driven to cut off the working places, shown by the dotted lines. The stables are located as shown in the shaft pillar, between the two shafts, where they will not contaminate the air going into the mine, but will receive air fresh from the downcast and discharge it at once into the upcast current. This position also affords ready access from either shaft in case of accident, and for the handling of feed and refuse. The pumps may be located in any convenient position at the foot of the upcast. The shaft bottoms are driven 14 ft. wide nearly through the shaft pillar, and are continued 10 ft. wide north and south through the gob. The width of all other roads and subroads is made 8 ft.

METHODS OF MINING ANTHRACITE

A perfectly flat seam of anthracite is seldom found in America, and even where a portion of the seam may be found lying comparatively flat, such sudden changes in dip must be expected that a system adapted to working on a pitch is almost universally used. A breast may start on a low pitch and the pitch may increase gradually until it becomes vertical, or the reverse

may be the case. The cleat is usually lacking in anthracite, and the direction of driving the breasts is determined largely by the pitch and by haulage considerations.

For pitches up to 30°, the methods already described are, in general, applicable, with certain changes due to local considerations. There is considerable difference in the methods of opening rooms in anthracite and bituminous mines, owing to variations in the characteristics of the coals and to the fact that anthracite will slide on chutes of less inclination than bituminous coal. Where the pitch does not exceed 4°, the rooms are turned off at right angles to the gangway. In moderately thick coal seams, pitching between 4° and 18°, the rooms are generally driven across the pitch, forming room breasts, thus securing a grade that permits the haulage of the cars to the face.

There are two methods of mining thick coal in breasts when nearly flat: (1) The breasts are opened out and driven to the limit in the lower bench of coal, and the top benches are blown down afterwards, beginning at the face and working back. (2) When the roof is good and there is no danger of its falling and closing up the workings, the upper benches may be worked in the opposite direction, beginning at the gangway and driving toward the limit of the lift, or the working of the upper bench may follow up that of the lower bench. When the seam is less than 12 ft., the top is supported by props; in thicker seams, the expense is so great for propping that but little attempt is made to support the roof. In the thicker anthracite seams (notably the Mammoth), the coal in the breasts is so worked as to make an arch of the upper benches of coal, which acts as a temporary support for the roof, the coal in the arch being extracted when the pillars are robbed.

When the inclination of anthracite seams is less than 30°, the breasts may be opened with one chute in the center, which ends in a platform projecting into the gangway, off which the coal can be readily loaded into the mine car. When this method is employed, the refuse is thrown to either side of the chute. If the pillars are to be robbed by skipping or slabbing one rib only, most of the refuse is kept on one side. Sometimes, when the top is good, and the breasts are driven wide, two chutes are used, but the cost of making the second chute is considerable

and is therefore not advisable unless necessitated by the method of ventilation employed.

Fig. 1 shows a panel system that gives good results in thick seams pitching from 15° to 45°, where the top is brittle, the coal free, and the mine gaseous. Rooms or breasts are turned off the gangway in pairs, at intervals of about 60 yd. The breasts are about 8 yd. wide, and the pillar between about 5 yd. wide, which is drawn back as soon as the breasts reach the airway, near the level above. In the middle of each large pillar between the several pairs of breasts, chutes about 4 yd. wide are driven from the gangway up to the airway above. These are provided with a traveling way on one side, giving the miners free access

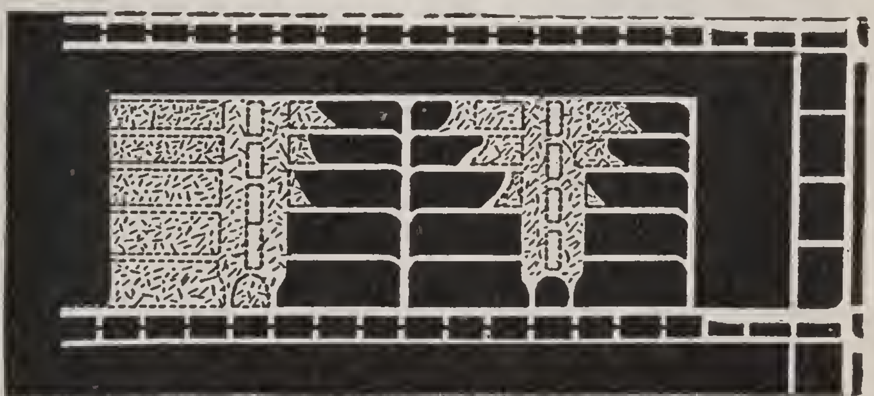


Fig. 1

to the workings. Small headings are driven in the bottom bench of coal, at right angles to these chutes, and about 10 or 20 yd. apart. These headings are continued on either side of the chutes until they intersect the breasts. When the chute and headings are finished, the work of getting the coal in the panel is begun by going to the end of the uppermost heading and widening it out on the rise side until the airway above is reached and a working face oblique to the heading is formed. This face is then drawn back to the chute in the middle of the panel. After the working face in the uppermost section has been drawn back some 10 or 12 yd., work in the next section below is begun, and so on down to the gangway. Both sides of the pillar are worked similarly and at the same time toward the chute.

Small cars, or buggies, are used to convey the coal from the working faces along the headings to the chute, where it is run down to the gangway below and loaded into the regular mine cars. This system affords a great degree of safety to the workmen, because whenever any signs of a fall of roof or coal occur, the men can reach the heading in a very few seconds and be perfectly safe. A great deal of narrow work must be done before any great quantity of coal can be produced. The breasts are driven in pairs and at intervals, to get a fair quantity of coal while the narrow work is being done, and they are not an essen-

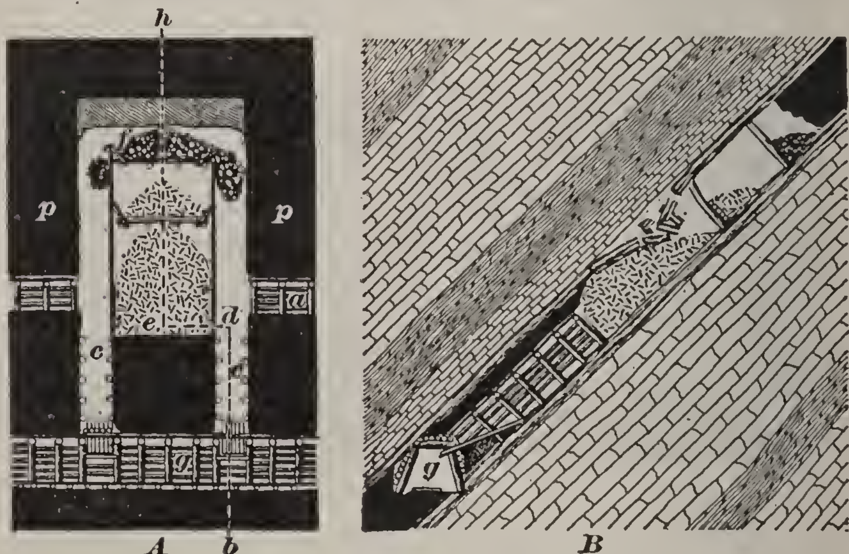


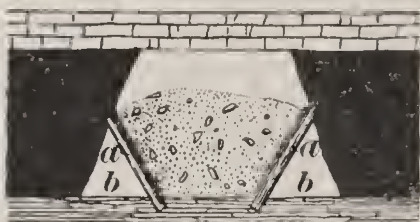
FIG. 2

tial part of the system. It is claimed that the facility and cheapness with which the coal can be mined, handled, and cleaned in the mine more than counterbalance the extra expense for the narrow work.

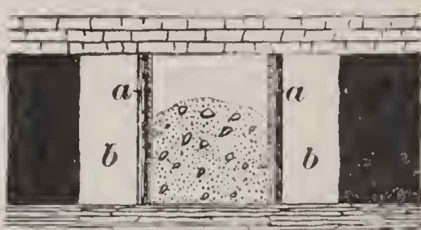
Battery Working.—Fig. 2 shows a method of opening a breast by two chutes *c*, *c*, when there is a great amount of refuse, or when a great amount of gas is given off. The chutes are extended up along the rib to within a few feet of the working face, either by planking carried on upright posts, or by building a jugular manway, as shown in Fig. 3, (*a*) and (*b*). These chutes, built of *jugulars* or inclined props and faced by 2-in.

plank, are made as nearly air-tight as possible, to carry the air from the heading *a* to the working face. Fig. 2 shows a breast on a pitch too steep to enable the miner to keep up to the face. In seams of less than 35°, the platform *f* shown near the face of the breast is unneccessary, and in seams thicker than 12 ft. it cannot be built; hence, this method of working is applicable only to beds pitching more than 35°, and to thin seams.

The coal is separated from the refuse on the platform *f*, and is run down the manway chutes and loaded into the cars from a platform projecting into the gangway *g*. The refuse is thrown in the middle of the breast behind the platform. A certain amount of coal is kept on the platform to deaden the blow from the falling coal. The chutes are timbered when the character of the coal requires it. This plan can also be employed in thick seams having a heavy dip, if there is enough refuse to fill the center of the breast so that the miner can work without the platform.



(a)



(b)

FIG. 3

Fig. 3 (*a*) is a section through *p*, Fig. 2, when jugulars *a*, are used to form the manways *b*, along the sides of the breast; and (*b*) is a section through the same line when upright posts *a* are used to support the plank in forming the manways *b*. The refuse in these cases only partially fills the gob.

In working very thick seams on heavy dips, where there is not enough refuse to fill the middle of the breast, the miner has nothing to stand on, the platform being impracticable; therefore, it is necessary to leave the loose coal in the breast. Loose coal occupies from 50% to 90% more space than coal in the solid. This surplus is drawn out through a central chute. If the roof is poor, the movement of the coal will not in this way cause it to fall and mix with the coal; and if the floor is soft,

the jugulars, which are stepped into the floor, are not so liable to be unseated, closing the manway and blocking the ventilation. The surplus is sometimes sent down the manways, leaving the loose coal in the center of the breast undisturbed, until the limit is reached.

Single-Chute Battery.—To prevent the coal from running out through the chutes, the opening into the breast is closed by a battery constructed by laying heavy logs across the openings, as shown at *b*, Fig. 4, or else built on props, as shown at *b*, Fig.

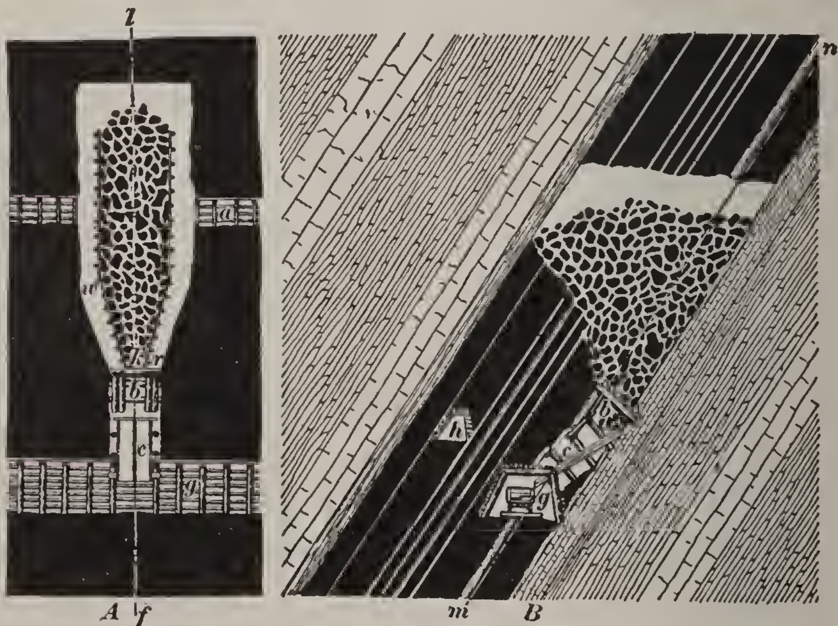


FIG. 4

5; a hole is left in the center, or at one side of the battery, through which the coal may be drawn. The battery closes all the openings into the breast, except the space occupied by the jugular manways, and is made air-tight, or as nearly so as possible, by a covering of plank.

Fig. 4 is a plan and section of a breast opened up by a single chute. The plan *A* is taken on the line *m n* shown on the section *B*, which section is taken on the line *f l* shown on the plan *A*. The pitch is great and the seam is so thick that the breast must be kept full of loose coal for the men to work upon,

the surplus being drawn off at the battery *b* and run into the car standing on the gangway *g* through the chute *c*. A manway *w* is made along each side of the breast, for the purpose of ventilation and affording a passage for the men to reach the working face. The heading *a* is used for an air-course between breasts. The main airway *h* is driven over the gangway *g*, where it will be well protected.

By drawing the surplus coal through a central chute, the manways are not injured so much as when it is drawn off

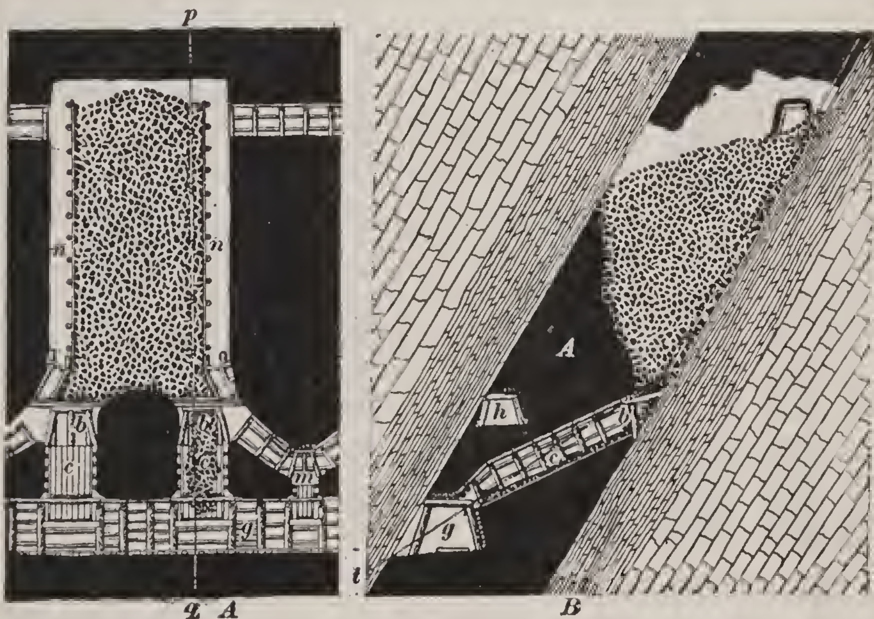


FIG. 5

through side chutes, as the coal will move principally along the middle of the breast. When the breast is worked up to its limit, all the loose coal is run out of the breast and the drawing back of the pillars is commenced, unless for some purpose they are allowed to stand for a time.

Double-Chute Battery.—Fig. 5 shows a plan and section of double-chute breasts used in very thick seams having a heavy dip. The breasts are entered by two main coal chutes *c*, each of which is provided with a battery *b*, through which the coal is drawn. A manway chute *m* is driven up through the middle of

the pillar for a few yards and is then branched in both directions until each branch (slant chute) intersects the foot of a breast near the battery *b*, as shown. The jugular manways *n*, are started at this point and continued up each side of the breast. The main airway *h* is driven in the solid through the stump *A* above the gangway. By driving the main gangway *g* against the roof, as shown, the pitch of the chute is lessened, and the loading chute *c* is more readily controlled.

When the main gangway is not driven against the roof, a gate is placed in the chute below the check-battery, which enables the loader to properly handle the coal. Coal in excess of the amount necessary to keep the miner up to the face may be drawn through the main battery, or sent down the manway chute, from which it is loaded through an air-tight check-battery.

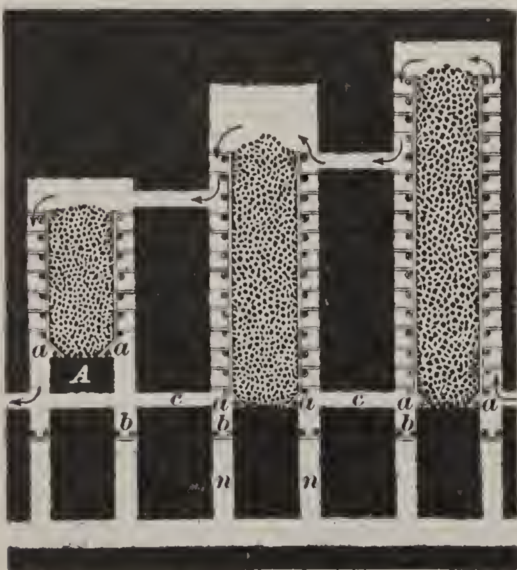


Fig. 6

The main chutes are usually 8 or 9 ft. wide, but sometimes only for the first 6 or 8 ft.; above this they are driven about 6 ft. square. The manway and *slant chutes* are also about 6 ft. square, which makes an easy passage.

When the seam is not thick enough to carry the return airway *h*, Fig. 5, over the gangway, the chutes are driven up in the manner there shown for a distance of about 30 ft., where they intersect the airway. The breast is opened out just above the airway, a battery being built in the airway immediately above each chute. A manway is driven from the gangway up through the middle of the stump until it intersects the airway, and a trap door is placed at this point to confine the air. This manway is made about 4 ft.×6 ft., or smaller, according to the amount of air needed.

Fig. 6 shows a less complicated plan than Fig. 5. The main chutes *n*, are driven up to the heading *c*, from which the breast is opened out; a log battery is built at the top of each chute at the points marked *a*. The chutes are used for drawing the battery coal, and for receiving the manway coal, and are also used for traveling ways. A check-battery *b* is placed in the chute to prevent the air-current from taking a short cut from the gangway through the chute to the breast airways. This check-battery is of great assistance to the loader when the chute has a very steep pitch, as he can readily control the flow of coal through the draw-hole.

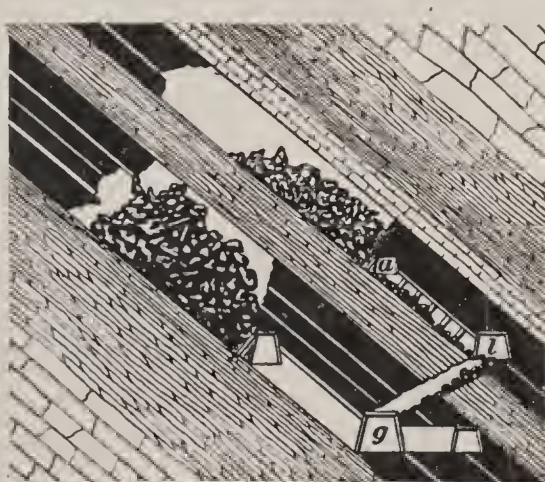


Fig. 7

All these methods are open to the objection that in case of any accident to the breast manway, by which the flow of air, shown by the arrows, is obstructed, there is no means of isolating the breast in which the accident occurs, and the ventilation of all the breasts beyond it is entirely stopped. To overcome this, sometimes the pillar *A*, shown in left-hand breast, in Fig. 6, is left in each breast to protect the airway.

Rock-Chute Mining.—Fig. 7 shows a section of two seams, separated by a few yards of rock. Chutes from $4\frac{1}{2}$ to 7 ft. high and 7 to 12 ft. wide are driven in the rock from the gangway or level *g* to the level *l* in the seam above, at such an angle

that the coal will gravitate from the upper seam into the gangway g. The working, otherwise, is similar to that previously described.

Fig. 8 shows how one or more seams are worked by connecting them by a *stone drift*, or *tunnel*, driven horizontally across the measures, through which the coal from the adjacent seams is taken to the haulage-way leading to the landing at the foot of the slope or shaft. Tunnels are sometimes driven horizontally

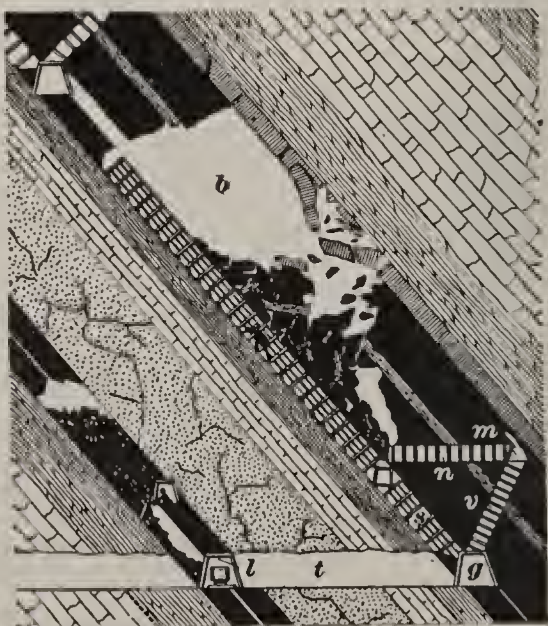


FIG. 8

through the measures from the surface, so as to cut one or more seams above water level.

The lower seam of coal is worked from a gangway or level *l*, connected by a tunnel, or stone drift *t*, to the level or gangway *g*, in the thick seam. The stone drift may be extended right and left to open seams above and below the thick seam. This tunnel, or stone drift, is never driven under a breast in the upper seam, but directly under the middle of the pillar.

In the upper and thicker seam, when the coal is very hard, a breast *b* is worked to the limit and the loose coal nearly all run

out through the chute *s* into the gangway *g*. The *monkey gangway m* is driven near the top as a return airway, and is connected to the upper end of the chute *s* by a level heading *n*, and to the main gangway *g* by a heading *v*. These headings are driven for the purpose of ventilation and to provide access to the battery in case the chute *s* should be closed. In the lower seam, the breast is still being worked upwards in the ordinary way.

FLUSHING OF CULM

From 15% to 20% of the coal taken out of an anthracite mine, according to the methods used in the past, became so fine in the course of preparation through the breaker that it could not be used or sold, and had to be piled away as refuse. Recently the coarser portions of these culm piles have been screened out and sold for use as steam sizes, while the finer part, together with the fine material from the breaker, has been carried back into the mines with water to fill the abandoned portions of the underground workings.

This culm is carried through a system of conveyors to the hopper, usually an old oil barrel, and the stream of water is conducted into the same hopper by a 3-in. pipe. The culm is then carried by the water through a pipe from 4 to 6 in. in diameter, which passes into the mine through the shaft, bore hole, or other opening, thence along the gangways to the chambers through the cross-cuts, and to the point where it is desired to deposit the culm. The bottoms or outlets of the chambers to be filled are closed by board partitions fitted closely, or by walls of slate or mine rubbish. The culm, as it issues from the end of the pipe, takes a very flat slope, and it is carried a long distance by the water, which ultimately filters through the deposited culm to the lower portion of the mine, to be pumped to the surface. When the chamber is filled to the roof, the pipe is withdrawn and extended to the next place to be filled, and so on.

The amount of water used depends on the distance to which the culm is carried and the slope of the pipe. From $1\frac{1}{2}$ to $1\frac{3}{4}$ lb. of water is required to flush 1 lb. of culm to level and downhill places; 3 to 6 lb. of water to 1 lb. of culm to flush up-hill

for heights varying from 10 to 100 ft. above the level of the shaft bottom. Any elevation of the pipe very materially increases the amount of water necessary.

To remove the pillars after the intervening breasts have been filled with culm, the face of the pillar along the gangway is attacked, and a road driven up through the pillar, splitting it, as shown in the accompanying illustration. This road may be the full width of the pillar, but in general it is necessary to leave the narrow stump of coal on either side to keep up the fine flushed material in the adjoining breasts. The thickness of this supporting coal depends entirely on the condition of the flushed material behind it.

EXPLOSIVES

Explosives are divided into two general classes: *Low explosives* or direct-exploding materials, and *high explosives* or indirect-exploding materials that require a detonator. The characteristics of a good blasting explosive are: sufficient stability and strength, difficulty of detonating by mechanical shock, handy form, absence of injurious effects on the user. Gunpowder or black powder is a low explosive; its composition depends on the purpose for which it is to be used, but the ingredients commonly used are saltpeter, sulphur, and charcoal. The high explosives are a mixture of nitroglycerine with an absorbing material, the composition of which is such that, in addition to thoroughly and permanently absorbing the nitroglycerine, it is itself a gas-producing compound.

Safety explosives, or as they are called, *permissible explosives*, are compounds intended for use in gaseous mines, and they are so constituted that they will ignite without producing the extremely high temperature given by ordinary explosives.

Permissible explosives may be arranged in four classes, the classification being based on the nature and proportions of the substances used in the manufacture. These classes are *hydrated, ammonium-nitrate, organic nitrate*, and *nitroglycerine explosives.* Permissible explosives are made by nearly all the manufacturers of blasting powder and a list is published each year by the United States Bureau of Mines giving the powders that have passed the test for permissible explosives with full instructions how they should be used and where manufactured.

Charging Explosives.—No invariable rule can be laid down as to the diameter and length of cartridges to be used under any and all circumstances, nor the amount or grade of powder required for all kinds of work. Much depends on the good sense and judgment of the persons using the explosives. In blasting coal, slate, marble, granite, freestone, or any other material that it is desirable to obtain in large blocks, cartridges of small diameter should be used in wide bore holes, the charge being rolled in several folds of paper, to prevent its touching the sides of the bore holes. The intensity of action and the crushing effect of the explosive are thus lessened. The charge must fit and fill the bottom of bore and be packed solid. If holes are comparatively dry, slit the paper of the cartridges lengthwise with a knife, and as each is dropped into the hole, strike a wooden rammer on it with sufficient force to make the powder completely fill the bottom and diameter of the bore. Where water is not present, a more perfect loading is made by taking powder out of cartridge and dropping it in loosely, ram each 6 or 8 in. of the charge, using the paper of each cartridge as a wad, to take down any powder that may have stuck to the sides of the hole. If water is standing in the hole, do not break the paper of the cartridges and avoid ramming more than enough to settle the charge on the bottom, using cartridges of as large diameter as will readily run into the bore.

When cartridges are used, the last cartridge placed in the hole should contain an electric exploder, or cap with fuse attached. When loose powder is used, a piece of cartridge 2 or 3 in. in length, with exploder or cap attached, should be pressed firmly on top of charge. Some blasters put an exploder or cap in the first cartridge used, placing remainder of charge on top.

If a seam is found in the rock, remove the powder from the cartridges and push it into the seam and fire a cap beside it. This will open the seam so that a larger quantity of explosive can be used, and the rock broken without drilling.

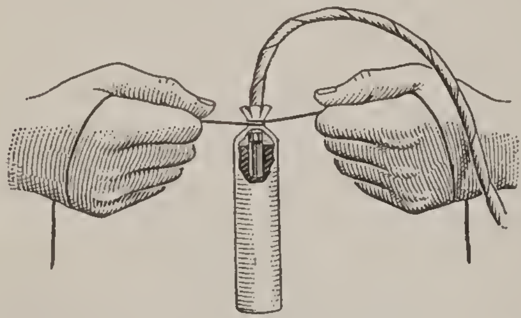


Fig. 1

Tamping. — In deep holes, water makes a good tamping, but fine sand, clay, etc. are generally used. Fill in for the first 5 or 6 in. carefully, so as not to displace cap and primer; then with a hardwood rammer pack balance of material as solid as possible, ramming with the hand alone,

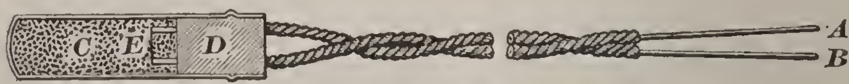


Fig. 2

and not using any form of hammer. Never use a metal tamping rod.

Firing.—If the work is wet, or the charge used under water, waterproof fuse must be used, and the end of the fuse protected by applying bar soap, pitch, or tallow around the edge of the cap. Water must not be allowed to reach the powder in the fuse or the fulminate in the cap. Exploding by electricity is best under water at great depth, as the pressure of water is so great that the water is forced through the fuse and it so prevents firing.

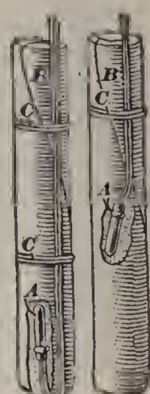


Fig. 3

Nitroglycerine explosives always require detonation by a cap or exploder in order to develop their full force. Fig. 1 illustrates the method of attaching such an exploder to the end of a fuse and the placing of it in the cartridge. The exploders are loaded with fulminate of mercury and are slipped over the end of the fuse, after which the upper end is crimped tightly

against the end of the fuse, as shown. (Miners sometimes bite the caps on the fuse with their teeth; this should never be allowed, for should one explode in a man's mouth it may prove fatal.) In placing the cap or exploder into the dynamite or giant-powder cartridge, care should be taken that only about two-thirds of the cap is embedded in the material of the cartridge, for if the fuse has to pass through a portion of the material before reaching the cap, it may ignite the material, thus causing deflagration of the cartridge in place of detonation. The fumes given off by high explosives are much worse in the case of deflagrating a cartridge.

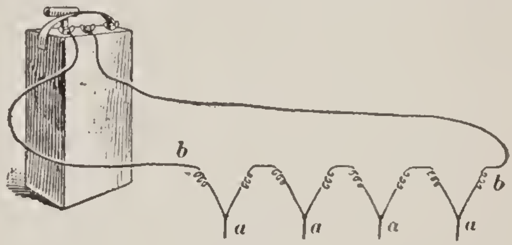


FIG. 4

The *electric exploder*, Fig. 2, consists of wires A and B that carry the current to the exploder; cement D (usually sulphur) that protects the explosive compound C (usually mercury fulminate), all of which is contained in a copper shell, and a small platinum wire E is heated by the passage of a current and ignites the explosive. Fig. 3 shows the method of placing a cap or an electric exploder in a cartridge of powder. When a number of holes are exploded at one time, the electric exploders are

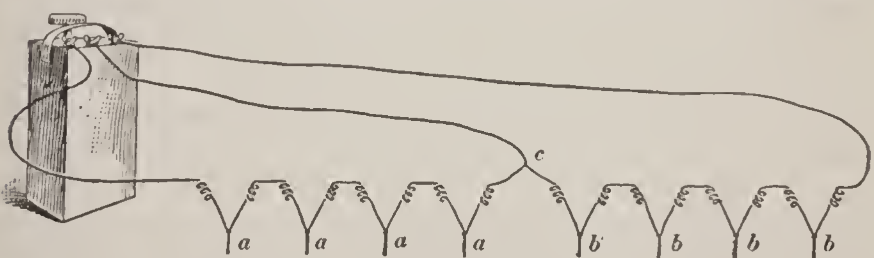


FIG. 5

connected in series, as shown in Fig. 4, for a small number of holes, and as in Fig. 5 for a larger number. The battery for furnishing the current of electricity is a magneto machine that is worked by pulling up or by depressing a handle or rack bar, or else by turning a crank.

Blasting by Electricity.—To blast by electricity, drill the number of holes desired to be fired at one time; the depth and spacing of the holes will depend on the character of rock, size of drill holes, etc., the blaster, of course, using his judgment in this matter. Load the hole in the usual manner, fitting one cartridge with a fuse (electric exploder) instead of cap and fuse. The fuse head is fitted into the bottom end of the cartridge, or into the middle of one side of the cartridge, where a hole has been punched with a pencil or small sharp stick to receive it; push the powder close around the fuse head. The fuse can then be held in position by tying a string around the cartridge and the fuse wires, binding the wires to the cartridge, as shown in Fig. 3, where *A* shows head of fuse, *B* the two fuse wires, *C* string used to tie wires to cartridge. Hitches should never be made in fuse wires, as the insulation of the wires may be injured and the current of electricity pass from one wire to the other, without passing through the cap, hazarding a misfire.

The cartridge containing the fuse is put in on top of the charge by some blasters; by others, at bottom of the charge. The best place for it is in the center of the charge. In tamping the hole, great care must be taken not to cut the wires, injure the cotton covering of fuse wires, or to pull the fuse out of the cartridge. At least 8 in. of the fuse wire should project above the hole, to make connections.

When all the holes to be fired at one time are tamped, separate the ends of the two wires in each hole, and, by the use of connecting wire, join one wire of the first hole with one of the second, the other or free wire of the second with one of the third, and so on to the last hole, leaving a free wire at each end hole. All connections of wires should be made by twisting together the bare and clean ends; it is best to have the joined parts bright. This may be done by scraping off the cotton covering at the ends of the wires to be connected, say for 2 in., then rubbing the wire with a small hard stone. All connections should be well twisted. Bare joints in wire should never be allowed to touch the ground, particularly if the ground is wet. This can be prevented by putting dry stones under the joints.

The charges having all been connected, the free wire of the first hole should be joined to one of the leading wires, and the

free wire of the last hole to the other of the two leading wires. The leading wires should be long enough to reach a point at a safe distance from the blast, say 250 ft. at least. All being ready, but not until the men are at a safe distance, connect the leading wires, one to each of the projecting screws on the front side or top of the battery, through each of which a hole is bored for the purpose, and bring the nuts down firmly on the wires. Take hold of the handle for the purpose, lift the rack bar (or square rod, toothed on one side) to its full length, and press it down, for the first inch of its stroke with moderate speed, but finishing the stroke with all force, bringing rack bar to the bottom of the box with a solid thud, and the blast will be made. Do not churn rack bar up and down. It is unnecessary and is harmful to the machine. One quick stroke of the rack bar is sufficient to make the blast. Never use fuses (exploders) made by different manufacturers in the same blast. Connecting wire should be of same size as the fuse wire; leading wire should be at least twice as large. Covering on wire should not strip or come off easily.

Arrangement of Drill Holes.—The arrangement of drill holes for driving and sinking should be such as to permit the easy handling of the drills and also to minimize the number of holes and the weight of the explosive. Two distinct systems are in use: the *center cut*, by which a center core or key is first removed, and after that concentric layers about this core; the *square cut*, in which the lines of holes are parallel to the sides of the excavation, the rock being removed in wedges instead of in concentric circles.

Thawing Dynamite.—All frozen cartridges should be thawed, as, when frozen, cartridges are very hard to explode, and even if they do explode, the results are not nearly as satisfactory as when properly thawed. When cartridges are frozen, they should not be exposed to a direct heat, but should be thawed by one of the following methods: (1) Place the number of cartridges needed for a day's work on shelves in a room heated by steam pipes (not live steam) or a stove; where regular blasting is done, a small house can be built for this purpose, fitted with a small steam radiator. Exhaust steam through these pipes gives all heat necessary. The house should be banked

around with earth, or, preferably, with fresh manure. (2) Use two water-tight kettles, one smaller than the other, put cartridges to be thawed in smaller kettle, and place it in larger kettle, filling space between the kettles with hot water at, say, 130° to 140° F., or so that it can be borne by the hand. To keep water warm, do not try to heat it in the kettle, but add fresh warm water. Cover kettles to retain heat. In thawing do not allow the temperature to get above 212° F. (3) Where the number of cartridges to be thawed is small, they may be placed about the person of the blaster until ready for use, the heat of the body thawing the cartridges.

MACHINE MINING

There are four general types of mining machines in use; pick machines, chain-cutter machines, cutter-bar machines, and longwall machines. The first two are the types almost universally used in America. Cutter-bar machines have almost entirely disappeared from use. Longwall mining machines have not been very generally adopted in America, as the longwall method of mining is not extensively used. Both compressed air and electricity are used for operating mining machines.

Pick machines work very similarly to a rock drill. They can be used wherever mining machines are applicable, and their particular advantage is that they are more perfectly under the control of the operator, who can cut around pyrites and similar obstructions without cutting them with the machine. This renders such a machine particularly applicable for seams of coal having rolls in the bottom and containing pyrites or other hard impurities. They are also applicable for working pillars on which there is a squeeze, as they are light and can be easily handled and readily removed.

A good pick machine will undercut 450 sq. ft. in 10 hr., while an ordinary miner will undercut 120 sq. ft. in the same time. In a seam varying from 4½ to 6 ft. in thickness, the machine will undercut from 50 to 100 T. of coal in 10 hr. The cost of undercutting under these conditions has been given as approximately 10c. per T. Extraordinary records show 1,400 sq. ft.

to have been cut in 9 hr. in Western Pennsylvania, and in an 8 ft. seam, 240 T. have been undercut in a shift of 10 hr.

Chain-cutter machines consist of a low metal bed frame upon which is mounted a motor that rotates a chain to which suitable cutting teeth are attached. To operate chain machines to the best advantage, the coal should be comparatively free from pyrites. They also require more room than pick machines, and a space from 12 to 15 ft. in width is necessary along the face to work them to advantage. A good chain cutter will make from 30 to 45 cuts, 44 in. wide and 6 ft. deep, in 10 hr. under moderately fair conditions, while in high seams two men handling the same machine under ordinary conditions can make 60 cuts per shift, and under particularly favorable conditions, 80 to 120 cuts per shift.

Shearing.—All the pick machines can be converted into *shearing machines* and can be used for longwall work by using a longer striking arm and a longer supply hose. The chain machines are used to do shearing work by having the cutting parts turned vertically.

VENTILATION OF MINES

COMPOSITION AND MEASUREMENT OF AIR

Air consists chiefly of oxygen and nitrogen, with small and varying amounts of carbonic-acid gas, ammonia gas, and aqueous vapor. These gases are not chemically combined, but exist in a free state in uniform proportion of 79.3% nitrogen and 20.7% oxygen by volume and 77% nitrogen and 23% oxygen by weight. Wherever air is found, its composition is practically the same. The weight of 1 cu. ft. of air at 32° F. and under a barometric pressure of 30 in. is .080975 lb. Air decreases in weight per cubic foot as its height above the sea level increases and it increases in weight below that level.

The weight of 1 cu. ft. of dry air at any temperature and barometric pressure is found by the formula

$$w = \frac{1.3253 \times B}{459 + t}.$$

in which w = weight of 1 cu. ft. of dry air;
B = barometric pressure, in inches of mercury;
t = temperature, in degrees F.

The constant 1.3253 is the weight in pounds avoirdupois of 459 cu. ft. of dry air at a temperature of 1° F. and 1 in. barometric pressure.

EXAMPLE.—Find the weight of 1 cu. ft. of dry air at a temperature of 60° F. and a barometric pressure of 30 in.

SOLUTION.—Applying the formula

$$\frac{1.3253 \times 30}{459 + 60} = .0766 \text{ lb.}$$

The *atmospheric pressure* is the pressure caused by the weight of the atmosphere above a given point. It is measured by the barometer, and this gives rise to the synonymous term *barometric pressure.* Atmospheric pressure is usually stated in pounds per square inch, while barometric pressure is stated in inches of mercury. Thus, at sea level, the atmospheric pressure under normal conditions of the atmosphere is 14.7 lb. per sq. in., while the barometric pressure at the same level is 30 in. of mercury column, or simply 30 in.

Mercurial Barometer.—The *mercurial barometer* is often called the *cistern barometer;* or, when the lower end of the tube is bent upwards instead of the mouth of the tube being submerged in a basin, it is known as the *siphon barometer.* The instrument is constructed by filling a glass tube 3 ft. long, and having a bore of $\frac{1}{4}$ in. diameter, with mercury, which is boiled to drive off the air. The thumb is then placed tightly over the open end, the tube inverted, and its mouth submerged in a basin of mercury. When the thumb is withdrawn, the mercury sinks in the tube, flowing out into the basin, until the top of the mercury column is about 30 in. above the surface of the mercury in the basin, and after a few oscillations above and below this point, comes to rest. The vacuum thus left in the tube above the mercury column is as perfect a vacuum as it is possible to form, and is called a *Torricelli vacuum*, after its discoverer.

Aneroid Barometer.— The *aneroid barometer* is a more portable form than the mercurial barometer. It consists of a

brass box resembling a steam-pressure gauge, having a similar dial and pointer, the dial, however, being graduated to read inches, corresponding to inches of mercury column, instead of reading pounds, as in a pressure gauge. Within the outer case is a delicate brass box having its upper and lower sides corrugated, which causes it to act as a bellows, moving in and out as the atmospheric pressure on it changes. The air within the box is partially exhausted, to render it sensitive to atmospheric changes. The movement of the upper surface of the box is communicated to the pointer by a series of levers, and the dial is graduated to correspond with the mercurial barometer.

Calculation of Atmospheric Pressure.—The weight of the mercury column of the barometer is the exact measure of the pressure of the atmosphere, since it is the downward pressure of the atmosphere that supports the mercury column, area for area; that is, the pressure of the atmosphere on 1 sq. in. supports a column of mercury having an area of 1 sq. in., and whose height is such that the weight of the mercury column is equal to the weight of the atmospheric column. Hence, as 1 cu. in. of mercury weighs .49 lb., the atmospheric pressure that supports 30 in. of mercury column is $.49 \times 30 = 14.7$ lb. per sq. in. In like manner, the atmospheric pressure corresponding to any height of mercury column may be calculated. The sectional size of the mercury column is not important because it is supported by the atmospheric pressure on an equal area, but the calculation of pressure is based on 1 sq. in.

Water Column Corresponding to Any Mercury Column.—The density of mercury referred to water is practically 13.6; hence, the height of a water column corresponding to a given mercury column is 13.6 times the height of the mercury column. For example, at sea level, where the average barometric pressure is 30 in. of mercury, the height of water column that the atmospheric pressure will support is $13.6 \times \frac{30}{12} = 34$ ft. This is the theoretical height to which it is possible to raise water by means of a suction pump, but the length of the suction pipe should not exceed 75% or 80% of the theoretical water column.

Finding Depth of Shafts.—The barometer is often used to determine the depth of a shaft or the depth of any point in a

mine below a corresponding point on the surface. The aneroid is used for this work, being more portable. Allowance must always be made in such cases for the ventilating pressure of the mine. A simple formula often used for such calculations is the following:

$$H = 55{,}000\left(1 - \sqrt{\frac{r}{R}}\right),$$

in which H = difference of level between two stations, in feet;
r = reading of barometer at higher station, in inches;
R = reading of barometer at lower station, in inches.

The most important use of the barometer in mining practice, however, is found in the warning that it gives of the decrease of atmospheric pressure, and the cxpansion of mine gases that always follows.

GASES FOUND IN MINES

Oxygen, *O*, is a colorless, odorless, tasteless, non-poisonous gas. It is heavier than air, having a specific gravity of 1.1056. It is the great supporter of life and combustion.

Nitrogen, *N*, is a colorless, odorless, and tasteless gas; it is neither combustible nor a supporter of combustion. It is not poisonous, and is lighter than air, having a specific gravity of .9713. Nitrogen is a particularly inert gas; it takes no active part in any combustion, in the sense of causing such combustion. Its province is to dilute oxygen of the atmosphere, on which life depends.

Methane, CH_4, often called *light carbureted hydrogen*, or marsh gas, is a chemical compound, consisting of 4 atoms of hydrogen to 1 atom of carbon. It is onc of the chief gases occluded in coal seams, and results from the mctamorphism of the carbonaceous matter from which coal is formcd, when such mctamorphism has taken place with thc exclusion of air, and in presence of water. Pure methane is colorless, odorless, and tasteless, and is lighter than air. Its specific gravity is .559, and it diffuses rapidly in the air, forming a firedamp mixture. Marsh gas burns with a blue flame, but it will not support combustion, and a lamp placed in it is immediately extinguishcd

Carbon monoxide, *CO*, often called *carbonic oxide gas*, or *white-damp*, is a chemical compound consisting of 1 atom of carbon

united to 1 atom of oxygen. To a certain extent it occurs as an occluded gas in coal. It is chiefly formed, however, in coal mines, by the slow combustion of carbonaceous matter in the gobs or waste places of the mine; where the supply of air is limited. It is always the product of the slow combustion of carbon in a limited supply of air, and is therefore one of the chief products of gob fires; it is also a product of the explosion of powder in blasting. This gas often fills the crevice made behind a standing shot, and causes the flash that takes place when the miner puts his lamp behind such shot to examine the same. This gas is formed in large quantities whenever the flame of a blast or explosion is projected into an atmosphere in which coal dust is suspended. The force of a blast often blows the dust into the air, and the flame acting on it distils carbon monoxide.

Carbon monoxide is lighter than air, having a specific gravity of .967, and it therefore accumulates near the roof and in the higher working places. It is colorless, odorless, and tasteless, but is combustible, burning with a light-blue flame. It is the flame often seen over a freshly fed anthracite fire. Carbon monoxide is a very poisonous gas, and acts on the human system as a narcotic, producing drowsiness or stupor, followed by acute pains in the head, back, and limbs, and afterward by delirium. When breathed into the lungs, it absorbs the oxygen from the blood, or, in other words, poisons the blood.

Carbon monoxide is detected in mine workings by its effect on the flame of a lamp, which burns more brightly in the presence of the gas, and reaches upwards as a slim, quivering taper, having often a pale-blue tip that, however, is not readily observed.

Carbon dioxide, CO_2, often called *carbonic-acid gas*, *black-damp*, or *choke damp*, is a chemical compound consisting of 1 atom of carbon united to 2 atoms of oxygen. It is heavier than air, having a specific gravity of 1.529. It therefore accumulates near the floor or in the low places of the mine workings. It is always the result of the complete combustion of carbon, in a plentiful supply of air, and is a product of the breathing of men and animals, burning of lamps, or any other complete combustion. It is always present in occluded gases.

Carbon dioxide is a colorless, odorless gas, but possesses a peculiarly sweet taste, which may be detected in the mouth when it is inhaled in large quantities. It is not combustible, nor is it a supporter of combustion. Lamps are at once extinguished by it. It diffuses slowly into the atmosphere, and is a difficult gas to remove in ventilating. It is not poisonous, but suffocates by excluding oxygen from the lungs. Its effect, when breathed for any length of time, is to cause headache and nausea, followed by weakness and pains in the back and limbs; when present in larger quantities, it causes death by suffocation. This gas, when present in the firedamp mixtures, has the opposite effect from that of carbon monoxide, inasmuch as it narrows the explosive range of the firedamp, and renders such mixtures inexplosive, which would otherwise be explosive.

Carbon dioxide is detected in the mine air by the dimness of the lamps and by their extinguishment when the gas is present in larger quantities. It should always be looked for at the floor, and in low places of the mine workings.

Hydrogen sulphide, H_2S, occurs at times as an occluded gas in coal seams, but more often exudes from the strata immediately underlying or overlying those seams. It is generally supposed to be formed by the disintegration of pyrites in the presence of moisture. It is heavier than air, having a specific gravity of 1.1912. It is a colorless gas, having a very disagreeable odor resembling that of rotten eggs, and is known to the miners as *stinkdamp*. It is an exceedingly dangerous gas when occurring in considerable quantities. When mixed with 7 times its volume of air, it is violently explosive. It is extremely poisonous, acting to derange the system, when breathed in small quantities, and, when inhaled in larger quantities, it produces unconsciousness and prostration. Its smell serves as the best means for its detection.

The general term *firedamp* relates to any explosive mixture of marsh gas and air, although in some localities this term is understood as referring to any mixture of methane and air whatever, whether explosive or otherwise. Many persons speak of pure methane as firedamp. The first meaning given, however, is the general acceptance of the term.

The term *afterdamp* relates to the gaseous mixture that exists in mine workings after an explosion of gas or coal dust. The composition of afterdamp is exceedingly variable, and admits of no general analysis that can be applied with certainty to any one explosion. The conditions that obtain in an explosion are so manifold, and control so completely the character of the gases formed, that it is impossible to give more than a general analysis of afterdamp.

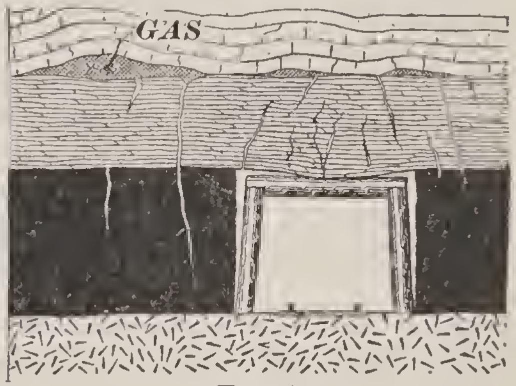


FIG. 1

Outbursts of gas are frequent occurrences in some coal seams. They are caused by the occluded gas finding its way to a vertical crevice or cleat in the coal seam, as illustrated in Fig. 1, and the pressure of the gas thus becomes distributed over a large area.

Testing for Gas by Lamp Flame.—Methane and firedamp are detected in mine workings by the small flame cap that envelopes and surmounts the flame of the lamp in a firedamp mixture.

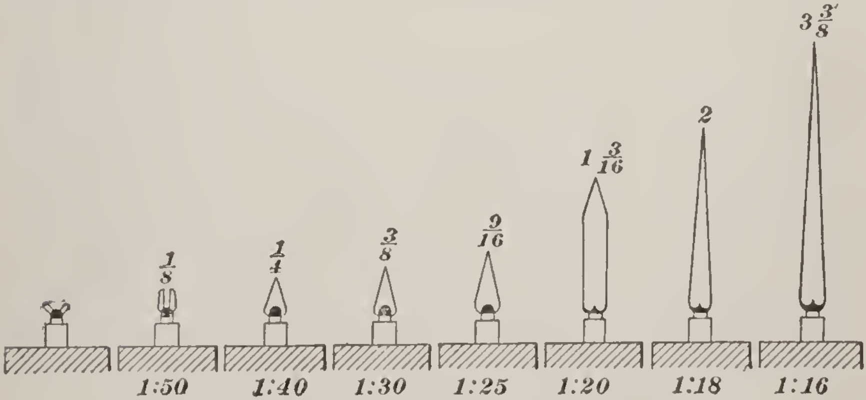


FIG. 2

This flame cap is caused by the gaseous mixture, which burns as it comes in contact with the flame. The proportion of gas in the mixture determines the height of the flame cap.

In Fig. 2, the heights of flame cap due to the presence of different proportions of methane are shown. These heights, as given, refer to the experimental heights of flame cap obtained with pure methane. It should be observed, however, that the presence of other gases in the firedamp will vary its explosive character, and this fact very materially modifies the explosiveness of certain caps.

Constants for Mine Gases.—The following table shows the symbols, specific gravities, and relative velocities of diffusion and transpiration of the principal mine gases, arranged in the order of their specific gravities, air being taken as 1. The values given in the next to the last column were obtained by experimenting with the gases, and agree quite closely with the calculated values given in the preceding column. This column shows that 1,344 volumes of methane will diffuse in the same time as 1,000 volumes of air, or 812 volumes of carbon dioxide.

TABLE OF MINE GASES

Name of Gas	Symbol	Specific Gravity	$\frac{1}{\sqrt{\text{sp. gr.}}}$	Relative Velocity of Diffusion (Air = 1)	Relative Velocity of Transpiration (Air = 1)
Air		1.00000	1.0000	1.000	1.0000
Carbon dioxide	CO_2	1.529	.8087	.812	1.2371
Hydrogen sulphide	H_2S	1.1912	.9162	.95	
Oxygen	O	1.1056	.9510	.9487	.903
Olefiant	C_2H_4	.978	1.0112	1.0191	1.788
Nitrogen	N	.9713	1.0147	1.0143	1.0303
Carbon monoxide	CO	.967	1.0169	1.0149	1.034
Steam	H_2O	.6235	1.2664		
Methane	CH_4	.559	1.3375	1.344	1.639
Hydrogen	H	.06926	3.7794	3.83	2.066

SAFETY LAMPS

The *safety lamp* is designed to give light in gaseous workings without the danger of igniting the gases present in the atmosphere. Its principle depends on the cooling effect that an iron-wire gauze exerts on flame. It is well known that all

gases ignite at certain fixed temperatures, and if this temperature is decreased from any cause, the flame is extinguished. Safety lamps are also used for testing for gas.

The essential features of a lamp designed for general mine work are: safety in strong currents, good illuminating power, security for lock fastening, freedom from flaming, security against accident, simplicity of construction. The essential features of a lamp for testing purposes are: free admission of air below the flame, no reflecting surface behind the flame, ability to test for a thin layer of gas at the roof. The Davy lamp in the hands of a careful person may be made to detect the presence of gas in quantities as low as 3%. It is claimed by some fire-bosses that 2% of gas may be detected with a good Davy. For the detection of small quantities of gas, especially constructed lamps have been used.

Types of Safety Lamps.—In the year 1815, Sir Humphrey Davy and George Stevenson, the latter a poor miner, discovered simultaneously, that flame would not pass through small openings in a perforated iron plate. This led to the construction of what are known as the Davy and the Stevenson or "Geordy," lamps. The Davy lamp is still a great favorite among fire-bosses for the detection of gas in mine air. Inasmuch as all safety lamps, of which there are a large number, depend on the same principle, only such lamps as possess essential features, and show important improvements and the gradual developments in safety-lamp construction are here mentioned. They are the Davy, Clanny, Mueseler, Marsant Ashworth-Hepplewhite-Gray, and Wolf.

Oils for Safety Lamps.—Most safety lamps burn vegetable oils, which are considered the safest for mining use; such oils are rape-seed oil and colzá oil, made from cabbage seed. Seal oil is also largely used. Seal oil affords a better light than vegetable oils, and in its use there is less charring of the wick. A mixture of 1 part of coal oil to 2 parts of rape or seal oil is often used and improves the light, but the smoke from the flame is increased. The Ashworth-Hepplewhite-Gray lamp is constructed to burn coal oil, or a mixture of coal and lard oil. The Wolf lamp is especially designed for burning naphtha or benzine. Special tests have been made to prove the safety

of using such a fluid in this lamp, and resulted in demonstrating the fact that the lamp was safe under any conditions that might arise. A thorough test was made, the oil vessel of the burning lamp being heated to 180° F., at which point the lamp was extinguished without manifesting any dangerous results.

Locking Safety Lamps.—The ordinary lock consists of a lead plug, which, when inserted in the lamp, will show the least tampering on the part of the miner. Other locks consist of an ordinary turnbolt operated by a peculiar key. Magnetic locks allow of the opening of the lamp only by means of a strong magnet kept in the lamp room.

Cleaning Safety Lamps.—Safety lamps should be thoroughly and regularly cleaned and filled between each shift. Each lamp should then be lighted and inspected by a competent person before being given to the miner. A careful inspection of the gauze of the lamp is necessary, as well as of all the joints by which air may enter the lamp. It should be known to a certainty that each lamp is securely locked before it leaves the lamp room.

Relighting Stations.—The relighting stations are located at certain places in gaseous mines where they can be supplied with a current of fresh air, and where there is no danger from the gases of the mine. The lamp is apt to be overturned, or to fall, and is often extinguished thereby; and if these stations were not provided, the man would have to return with his lamp to the surface in order to have it relighted. Such a station is always located at the entrance of the gaseous portion of a mine, in cases where the entire mine does not liberate gas. A number of self-igniters have been invented and some are used. If the lamp goes out all that is needed is to turn a screw in the bottom of the lamp and a spark is made which relights the lamp.

Illuminating Power of Safety Lamps.—The accompanying table gives the illuminating power or candlepower of some of the principal lamps. The light of a sperm candle is taken as 1, or unity.

Acetylene Mine Lamps.—Acetylene gas is generated by dropping water on to calcium carbide; as in the automobile and bicycle acetylene lamps. The miner's lamps using

acetylene are small brass lamps consisting of two main parts, the carbide container and the water tank; a regulator limits the amount of water dropping on to the carbide. The gas given off burns with a bright white flame. Besides giving greater illumination, acetylene lamps burn practically without generating soot, and are much less harmful to the miner's respiratory organs than the constantly smoking oil lamps. Ventilation is also facilitated, owing to the acetylene lamp consuming less oxygen than any other. Acetylene illumination is also cheaper than oil lighting.

Electric Mine Lamps.—Several forms of electric mine lamps have been invented. Some of them are in use by miners, but others are not extensively used, on account of their weight. These heavier types may be used by mine rescue parties. Electric mine lamps should have a battery that is not heavy, a good tungsten lamp, and a good strong reflector. In the Hirsch is an example which is carried by the miner, the battery is fastened on the miner's back to a belt passing around his waist. An insulated copper wire transmits the current from

LIGHT GIVEN BY SAFETY LAMPS

Name of Lamp	Illuminating Power of Lamp Candlepower
Davy	.16
Geordy	.10
Clanny	.20
Mueseler	.35
Evan Thomas	.45
Marsaut, 3 gauzes	.45
Marsaut, 2 gauzes	.55
Marsaut, with Howat's deflector	.65
Ashworth-Hepplewhite-Gray	.65
Wolf	.90

the battery to the lamp on the miner's cap. The battery is charged by connecting it with the current in the lamp house at night and is ready for use the next morning. A type of electric amps to carry in the hand and for use on locomotives is the Hubbell electric lantern

EXPLOSIVE CONDITIONS IN MINES

In the ventilation of gaseous seams, the air-current may be rendered explosive by the sudden occurrence of any one of a number of circumstances that cannot be anticipated. Hence, the condition of the air-current should be maintained far within the explosive limit. Among these are the following: (1) Derangement of the ventilating current; (2) sudden increase of gas due to outbursts, falls of roof, feeders, fall of barometric pressure, etc.; (3) Presence of coal dust thrown into suspension in the air, in the ordinary working of the mine, or by the force of blasting at the working face, or by a blown-out, or windy, shot; (4) Pressure due to a heavy blast, or any concussion of the air caused by closing of doors, etc.; (5) rapid succession of shots in close workings; (6) accidental discharge of an explosive in a dirty atmosphere. The explosive conditions vary considerably in different coal seams, as the nature of the coal and its enclosing strata, its friability and inflammability, together with the character of its occluded gases, determine, to a large extent, the explosive conditions. A great many of these conditions have been investigated by the U. S. Bureau of Mines and the results published in pamphlet form.

Mine Explosions.—The explosion of gas in a mine usually arises from the ignition of an explosive mixture of gas and air called *firedamp*, which has accumulated in some unused chamber or cavity of the roof, or in the waste places of the mine, and has been ignited by a naked light, by the flame of a shot, or by a mine fire. The initial force of an explosion is generally expended locally, but the flame continues to feed upon the carbon monoxide generated by the incomplete combustion of the firedamp mixture, and distilled also from the coal dust thrown into the air by the agitation. Air is required to burn this carbon monoxide; this causes the flame to travel against the air-current, or in the direction in which fresh air is found. In the other direction, or behind the explosion, the flame is soon extinguished in its own trail when the initial force of the explosion is expended. The explosion continues to travel along the airways against the current as long as there is sufficient gas or coal dust for it to feed upon, or until its

temperature is cooled below the point of ignition, by some cause such as, for example, the rapid expansion of the area of the workings. The chief factor in transmitting an explosion is the presence of carbon monoxide, which lengthens the flame and extends the effect.

The *recoil of an explosion* is the return of the flame along the path that it has just traversed. In the recoil, the flame burns more quietly, advances more slowly and travels close to the roof.

QUANTITY OF AIR REQUIRED FOR VENTILATION

The quantity of air required for the adequate ventilation of a mine cannot be stated as a rule applicable in every case. Regulations that would supply a proper amount of air for the ventilation of a thick seam would cause great inconvenience if applied without modification to the workings in a thin seam.

The quantity of air required by the laws of the several States is generally specified as 100 cu. ft. per min. per man and in many cases an additional amount of 500 cu. ft. per min. per animal is stated. This quantity is in no case stated as the actual amount of air required for the use of each man or animal, but is only the result of experience, as showing the quantity of air required for the proper ventilation of the average mine, based on the number of men and animals employed. The number of men employed in a mine is an indication of the extent of the working face, while the number of animals employed is an indication likewise of the extent of the haulage roads, or the development of the mine. These amounts refer particularly to non-gaseous seams.

The Bituminous Mine Law of Pennsylvania specifies that there shall be not less than 100 cu. ft. per min. per person in any mine, while 150 cu. ft. is required in a mine where fire-damp has been detected.

The Anthracite Mine Law of Pennsylvania specifies a minimum quantity of 200 cu. ft. per min. per person. Each of these laws contains modifying clauses, which specify that the amount of air in circulation shall be sufficient to "dilute, render harmless, and sweep away" smoke and noxious or dangerous gases.

ELEMENTS OF VENTILATION

The elements in any circulation of air are (1) horsepower, or power applied; (2) resistance of airways, or mine resistance, which gives rise to the total pressure in the airway; (3) velocity generated by the power applied against the mine resistance.

Horsepower or Power of Current.—The power applied is often spoken of as the *power upon the air*. It is the effective power of the ventilating motor, whatever this may be, including all the ventilating agencies, whether natural or otherwise. The power upon the air may be the power exerted by a motive column due to natural causes, or to a furnace, or may be the power of a mechanical motor. The power upon the air is always measured in foot-pounds per minute, which expresses the units of work accomplished in the circulation.

Mine Resistance.—The resistance offered by a mine to the passage of an air-current, or the mine resistance, is due to the friction of the air rubbing along the sides, top, and bottom of the air passages. This friction causes the total ventilating pressure in the airway, and is equal to it, or, the total pressure is equal to the mine resistance or

$$R = pa,$$

in which R = resistance;
p = unit of ventilating pressure;
a = sectional area of airway.

Velocity of Air-Current.—Whenever a given power is applied against a given resistance, a certain velocity results. For example, if the power u, in foot-pounds per minute, is applied against the resistance $p\,a$, a velocity of v, in feet per minute, is the result; and as the total pressure $p\,a$ moves at the velocity of v, the work performed each minute by the power applied is the product of the total pressure by the space through which it moves per minute, or the velocity. Thus, $u = (p\,a)v$.

Relation of Power, Pressure, and Velocity.—The relation of power, pressure, and velocity is not a simple one. For example, a given power applied to move air through an airway establishes a certain resistance and velocity in the airway. The resistance of the airway is not an independent factor; that is, it does not exist as a factor of the airway independent of the velocity, but bears a certain relation to the velocity. Power

always produces resistance and velocity, and these two factors always sustain a fixed relation.

This relation is expressed as follows: The *total pressure or resistance varies as the square of the velocity;* i. e., if the power is sufficient to double the velocity, the pressure will be increased 4 times; if the power is sufficient to multiply the velocity 3 times, the pressure will be increased 9 times. Thus, a change of power applied to any airway means both a change of pressure and a change of velocity.

Again, as the power is expressed by the equation $u=(p\ a)v$, and as $p\ a$, or the total pressure, varies as v^2, the work varies as v^3. Therefore, if the velocity is multiplied by 2, and, consequently, the total pressure by 4, the work performed $(p\ a)v$ will be multiplied by $2^3=8$, or the power applied varies as the cube of the velocity.

MEASUREMENT OF VENTILATING CURRENTS

The measurement and calculation of any circulation in a mine airway includes the measurement of: (1) the velocity of the air-current, (2) of pressure, (3) of temperature, (4) calculation of pressure, quantity, and horsepower of the circulation. These measurements should be made at a point on the airway where the airway has a uniform section for some distance, and not far from the foot of the downcast shaft or the fan drift.

Measurement of Velocity.—For the purpose of mine inspection, the velocity of the air-current should be measured at the foot of the downcast, at the mouth of each split of the air-current, and at each inside breakthrough, in each split. These measurements are necessary in order to show that all the air designed for each split passes around the face of the workings.

The measurement of the velocity of a current is best made by means of the anemometer.

Rule.—*To obtain the quantity of air passing in cubic feet per minute, multiply the area of the airway, at the point where the velocity is measured, by the velocity.*

Measurement of Pressure.—The measurement of the ventilating pressure is made by means of a water column in the form of a water gauge, which is simply a glass U tube open at

both ends, as shown in Fig. 3. Water is placed in the bent portion of the tube, and stands at the same height in both arms of the tube when each end of the tube is subjected to the same pressure. If, however, one end of the tube is subjected to a greater pressure than the other end, the water will be forced down in that arm of the tube, and will rise a corresponding height in the other arm, the difference of level in the two arms of the tube representing the water column balanced by the excess of pressure to which the water in the first arm is subjected. An adjustable scale, graduated in inches, measures

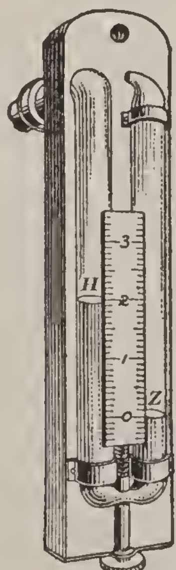


Fig. 3

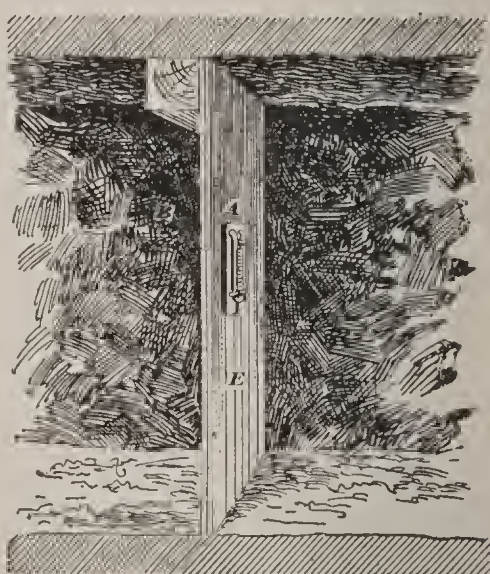


Fig. 4

the height of the water column. The zero is adjusted to the lower water level and the upper water level will then give the reading of the water gauge. One end of the glass tube is drawn to a narrow opening to exclude dust, while the other end is bent to a right angle, and passing back through the standard to which the tube is attached, is cemented into the brass tube that passes through a hole in the partition or brattice, when the water gauge is in use. The bend of the tube is contracted to reduce the tendency to oscillation in the height of water column.

When in use, the water gauge must be in a perpendicular position. It is placed upon a brattice occupying a position between two airways, as shown at *A*, Fig 4. The brass tube forming one end of the water gauge is inserted in a cork, and passes through a hole bored in the brattice. The water gauge must not be subjected to the direct force of the air-current, as in this case the true pressure will not be given. Fig. 4 shows the instrument occupying a position in the breakthrough, between two entries. It will be observed that the water gauge records a difference of pressure, each end of the water gauge being subject to atmospheric pressure, but one end in addition being subject to the ventilating pressure, which is the difference of pressure between the two entries. The water gauge thus permits the measurement of the resistance of the mine inbye from its position between two airways. If placed in the first breakthrough, at the foot of the shaft, it measures the entire resistance of the mine, but if placed at the mouth of a split, it measures only the resistance of that split. It never measures the resistance outbye from its position in the mine, but always inbye.

Measurement of Temperature.—It is important to measure the temperature of the air-current at the point where the velocity is measured, as the temperature is an important factor of the volume of air passing.

Calculation of Mine Resistance.—The mine resistance is equal to the total pressure pa that it causes. This mine resistance is dependent on three factors: (1) The resistance k offered by 1 sq. ft. of rubbing surface to a current having a velocity of 1 ft. per min. The coefficient of friction k, or the unit of resistance, is the resistance offered by the unit of rubbing surface to a current of a unit velocity. This unit resistance has been variously estimated by different authorities. The value most universally accepted, however, is that known as the Atkinson coefficient .0000000217. (2) The mine resistance, which varies as the square of the velocity. (3) The rubbing surface. Hence, if the unit resistance is multiplied by the square of the velocity, and by the rubbing surface, the total mine resistance as expressed by the formula $pa = ksv^2$, will be obtained.

Calculation of Power, or Units of Work per Minute.—If the total pressure is multiplied by the velocity in feet per minute, with which it moves, the units of work per minute, or the power upon the air, will be obtained. Hence, $u = pav = ksv^3$, which is the fundamental expression for work per minute, or power.

The Equivalent Orifice.—The term *equivalent orifice*, often used in regard to ventilation, evaluates the mine resistance, or, as will be seen from the equation given for its value, it expresses the ratio that exists between the quantity of air passing in an airway and the pressure or water gauge that is produced by the circulation. This term refers to the flow of a fluid through an orifice in a thin plate, under a given head. The formula expressing the velocity of flow through such an orifice is $v = \sqrt{2gh}$; multiplying both members of this equation by A, and substituting for the first member Av, its value q, after transposing and correcting $A = \frac{q}{.62\sqrt{2gh}}$, in which .62 is the coefficient for the contracted vein of the flow. Reducing this to cubic feet per minute and inches of water gauge represented by i, gives the equation $A = .0004 \times \frac{q}{\sqrt{i}}$. By this formula, Murgue has suggested assimilating the flow of air through a mine to the flow of a fluid through a thin plate, for in each case, the quantity and the head or pressure vary in the same ratio. Thus, applying this formula to a mine, Murgue multiplies the ratio of the quantity of air passing, in cubic feet per minute, and the square root of the water gauge, in inches, by .0004, and obtains an area A, which he calls the equivalent orifice of the mine.

Potential Factor of a Mine.—Ventilating formulas 8 and 27 page (267) give, respectively, the pressure and the power that will circulate a given quantity of air per minute in a given airway. These formulas may be written as equal ratios, expressed in factors of the current and the airway, respectively; thus, $\frac{p}{q^2} = \frac{ks}{a^3}$, and $\frac{u}{q^3} = \frac{ks}{a^3}$, which show that the ratio between the pressure and the square of the quantity it circulates in any given airway is equal

to the ratio between the power and the cube of the quantity it circulates. Solving each of these formulas with respect to q:

With respect to pressure,

$$q = \left(a\sqrt{\frac{a}{ks}}\right)\sqrt{p}$$

With respect to power,

$$q = \left(\frac{a}{\sqrt[3]{ks}}\right)\sqrt[3]{u}$$

Hence, in any airway, for a constant pressure, the quantity of air in circulation is proportional to the expression $a\sqrt{\dfrac{a}{ks}}$; and for a constant power, the quantity is proportional to the expression $\dfrac{a}{\sqrt[3]{ks}}$, which terms are called the potentials of the mine with respect to pressure and power, respectively; and their values $\dfrac{q}{\sqrt{p}}$ and $\dfrac{q}{\sqrt[3]{u}}$ are the potentials of the current with respect to pressure and power, respectively. These factors evaluate the airway, as they determine the quantity of air a given pressure or power will circulate in that airway, in cubic feet per minute. By their use, the relative quantities of air any given pressure or power will circulate in different airways are readily determined. The rule may be stated as follows:

Rule.—*For any given pressure or power, the quantity of air in circulation is always proportional to the potential for pressure, or the potential for power, as the case may be.*

This rule finds important application in splitting. In all cases where the potential is used as a ratio, the relative potential may be employed by omitting the factor k; or it may be employed to obtain the pressure and power, in several splits by multiplying the final result by k, as in the splitting formulas 46 and 47, page 275.

The accompanying table will illustrate the use of the formulas used in these calculations. There are several formulas for quantity, velocity, and work or horsepower, but in each case

the several formulas are derived by simple transposition of the terms of the original formulas $p=\frac{ksv^2}{a}$, $q=av$, and $u=qp$.

To illustrate the use of the formulas, take an underground road, 5 ft. wide by 4 ft. high, and 2,000 ft. in length, and calculate the value of each symbol or letter, assuming a velocity of 500 ft. per min.

	Symbol	Value of Symbol
Area of airway (5 ft.×4 ft.)	a	20 sq. ft.
Horsepower*	h	2.959 H. P.
Coefficient of friction†	k	.0000000217 lb.
Length of airway	l	2,000 ft.
Perimeter of airway, 2×(5 ft.+4 ft.)	o	18 ft.
Pressure, in pounds per sq. ft.	p	9.765 lb.
Quantity of air, in cubic feet per minute	q	10,000 cu. ft.
Area of rubbing surface	s	36,000 sq. ft.
Units of work per minute, power	u	97,650 ft.-lb.
Velocity, in feet per minute	v	500 ft.
Water gauge	i	1.87788 in.
Equivalent orifice of the mine	A	2,919 sq. ft.
Potential for power	X_u	217.16 units
Potential for pressure	X_p	3,200 units
Weight of 1 cu. ft. of downcast air	w	.08098 lb.
Motive column, downcast air	M	120.5 ft.
Depth of furnace shaft	D	306.77 ft.
Average temperature of the upcast column	T	350° F.
Average temperature of the downcast column	t	32° F.

NOTE.—The water gauge is calculated to 5 decimal places to enable all the other values to be accurately arrived at; in practice, it is only read to 1 decimal place.

*A horsepower is equal to 33,000 units of work.

†This coefficient of friction is an invariable quantity, and is the same in every calculation relating to the friction of air in mines.

VENTILATION FORMULAS

To Find	No.	Formula	To Find	No.	Formula
Rubbing surface of an airway in sq. ft.	1	$s = lo$	Water gauge in inches.	14	$i = \frac{p}{5.2}$
Area of an airway, in sq. ft.	2	$a = \frac{q}{v}$	Resistance of an airway in lb.	15	$pa = ksv^2$
Velocity in ft. per min.	3	$v = \frac{q}{a}$		16	$pa = \frac{u}{v}$
	4	$v = \sqrt[3]{\frac{u}{ks}}$	Quantity in cu. ft. per min.	17	$q = av$
	5	$v = \sqrt{\frac{pa}{ks}}$		18	$q = \frac{u}{p}$
	6	$v = \frac{u}{pa}$		19	$q = \sqrt{\frac{pa}{ks}} \times a$
Pressure in lb. per sq. ft.	7	$p = \frac{ksv^2}{a}$		20	$q = \sqrt[3]{\frac{u}{ks}} \times a$
	8	$p = \frac{ksq^2}{a^3}$		21	$q = X_u \sqrt[3]{u}$
	9	$p = \frac{u}{q}$		22	$q = \sqrt[3]{X_p^2 u}$
	10	$p = Mw$		23	$q = X_p \sqrt{p}$
	11	$p = 5.2\,i$	Units of work per minute, or power on air, in ft.-lb. per min.	24	$u = avp$
	12	$p = \frac{q^2}{X_u^3}$		25	$u = qp$
	13	$p = \frac{q^2}{X_p^2}$		26	$u = ksv^3$
				27	$u = \frac{ksq^3}{a^3}$
				28	$u = h33{,}000$

TABLE—(*Continued*)

To Find	No.	Formula	To Find	No.	Formula
Units of work per minute, or power on air, in ft.-lb. per min.	29	$u=\frac{q^3}{X_u{}^3}$	Pressure potential, in units	35	$X_p=a\sqrt{\frac{a}{ks}}$
	30	$u=\frac{q^3}{X_p{}^2}$		36	$X_p=\frac{q}{\sqrt{p}}$
Horse-power	31	$h=\frac{u}{33{,}000}$	Equivalent orifice, in sq. ft.	37	$A=\frac{.0004q}{\sqrt{i}}$
Power potential, in units	32	$X_u=\frac{a}{\sqrt[3]{ks}}$	Motive column, downcast air, in ft.	38	$M=D\times\frac{T-t}{459+T}$
				39	$M=\frac{p}{w}$
	33	$X_u=\sqrt[3]{\frac{q^2}{p}}$	Motive column, upcast air, in ft.	40	$M=D\times\frac{T-t}{459+t}$
	34	$X_u=\frac{q}{\sqrt[3]{u}}$		39	$M=\frac{p}{w}$

As specimen calculations, take a formula for pressure and one for quantity. Taking the values from the table and formula 7, $p=\frac{ksv^2}{a}$ for pressure and formula 20, $q=\sqrt[3]{\frac{u}{ks}}\times a$, gives—

$$p=\frac{.0000000217\times 36{,}000\times 500^2}{20}=9.765 \text{ lb.}$$

and

$$q=\sqrt[3]{\frac{97{,}650}{.0000000217\times 36{,}000}}\times 20=10{,}000 \text{ cu. ft.}$$

In this way any of these formulas may be worked out.

Variation of Elements.—In the foregoing table, fixed conditions of motive column, as well as fixed conditions in the mine airways, were assumed. It is often convenient, however, to

know how the different elements, as velocity v, quantity q, pressure p, power u, etc., will vary in different circulations; as, by this means it is possible to compare the circulations in different airways, or the results obtained by applying different pressures and powers to the same airway. These laws of variation must always be applied with great care. For example, before it is possible to ascertain how the quantity in circulation will vary in different airways, it is necessary to know whether the pressure or the power is constant or the same for each airway. The following rules may always be applied:

For a constant pressure: v varies as $\sqrt{\frac{a}{lo}}$; q varies as $a\sqrt{\frac{a}{lo}}$ (relative potential for pressure).

For a constant power: v varies as $\frac{1}{\sqrt[3]{lo}}$; q varies as $\frac{a}{\sqrt[3]{lo}}$ (relative potential for power).

For a constant velocity: q varies as a; p varies as $\frac{lo}{a}$; u varies as lo.

For a constant quantity: v varies inversely as a; p varies inversely as $X_u{}^3$ (potential for power); u varies inversely as $X_u{}^3$ (potential for power) or directly as p.

For the same airway: The following terms vary as each other: v, q, $\sqrt{p}$, $\sqrt[3]{u}$.

Similar Airways: r = length of similar side, or similar dimension.

For a constant pressure: v varies as $\sqrt{\frac{r}{l}}$; q varies as $r^2 \times \sqrt{\frac{r}{l}}$; r varies as lv^2, or $\sqrt[5]{lq^2}$.

For a constant power: v varies as $\frac{1}{\sqrt[3]{lr}}$; q varies as $r \times \sqrt[3]{\frac{r^2}{l}}$; r varies as $\frac{1}{lv^3}$, or $\sqrt[5]{lq^3}$.

For a constant velocity: q varies as r^2; p varies as $\frac{l}{r}$; u varies as lr; r varies as $\sqrt{q}$, $\frac{l}{p}$, or $\frac{u}{l}$.

For a constant quantity: v varies inversely as r^2; p and u vary inversely as $\frac{r^5}{l}$; r varies as $\frac{1}{\sqrt{v}}$, $\sqrt[5]{\frac{l}{p}}$, or $\sqrt[5]{\frac{l}{u}}$.

Furnace Ventilation.—p (motive column) varies as D; q varies as $\sqrt{D}$.

Fan Ventilation.—It has been customary in calculations pertaining to the yield of centrifugal ventilators to assume as follows: q varies as n; p varies as n^2; u varies as n^3.

More recent investigation, however, shows that doubling the speed does not double the quantity of air in circulation; or, in other words, the quantity does not vary exactly as the number of revolutions of the fan. Investigation also shows that the efficiency of centrifugal ventilators decreases as the speed increases. To what extent this is the case has not been thoroughly established. The variation between the speed of a fan and the quantity, pressure, power, and efficiency, as calculated from a large number of reliable fan tests, may be stated as follows:

For the same fan, discharging against a constant potential: q varies as $n^{.97}$; p varies as $n^{1.94}$. Complement of efficiency $(1-K)$ varies as $n^{.425}$.

The efficiency here referred to is the mechanical efficiency, or the ratio between the effective work qp and the theoretical work of the fan.

DISTRIBUTION OF AIR IN MINE VENTILATION

Legal Requirements.—The Anthracite Mine Law of Pennsylvania specifies that every mine employing more than 75 persons must be divided into two or more ventilating districts, thus limiting the number that are allowed to work on one air-current to 75 persons. The Bituminous Mine Law of Pennsylvania limits the number allowed to work upon one current to 65 persons, except in special cases, where this number may be increased to 100 persons at the discretion of the mine inspector.

Splitting of Air-Current.—When the air-current is divided into two or more branches, it is said to be *split*. The current may be divided one or more times; when split or divided once, the current is said to be traveling in *two splits*, each branch being termed a split. The number of splits in which a current is made to travel is understood as the number of separate currents in the mine, and not as the number of divisions of the current.

When the main air-current is divided into two or more splits, each of these is called a *primary split*. *Secondary splits* are the divisions of a primary split. *Tertiary splits* result from the division of a secondary split.

Equal Splits of Air.—When a mine is spoken of as having two or more equal splits, it is understood to mean that the length and the size of the separate airways forming those splits are equal in each case. It follows, of course, from this that the ventilating current traveling in each split will be the same, inasmuch as they are all subject to the same ventilating pressure, When an equal circulation is obtained in two or more splits by the use of regulators, these splits cannot be spoken of as equal splits.

Natural Division of Air-Current.—By natural division of air is meant any division of the air that is accomplished without the use of regulators; or, in other words, such division of the air-current as results from natural means. If the main air-current at any given point in a mine is free to traverse two separate airways in passing to the foot of the upcast shaft, and each of these airways is free or an open split, i. e., contains no regulator, the division of the air will be a natural division. In such a case, the larger quantity of air will always traverse the shorter split of airway. In other words, an air-current always seeks the shortest way out of a mine. A comparatively small current, however, will always traverse the long split or airway.

It is always assumed, in the calculation of the splitting of air-currents, that the pressure at the mouth of each split, starting from any given point, is the same. In order to find the quantity of air passing in each of several splits starting from a common point, the following rule may be applied.

Rule.—*The ratio between the quantity of air passing in any split and the pressure potential of that split is the same for all splits starting from a common point. Also, the ratio between the entire quantity of air in circulation in the several splits and the sum of the pressure potentials of those splits is the same as the above ratio, and is equal to the square root of the pressure.*

Stated as a formula, indicating the sum of the pressure potentials (X_1+X_2+etc.) by the expression ΣX_p,

$$\frac{Q}{\Sigma X_p} = \frac{q_1}{X_1} = \sqrt{p}$$

Hence, $p = \frac{Q^2}{(\Sigma X_p)^2}$ and $u = \frac{Q^3}{(\Sigma X_p)^2}$ express the pressure and power respectively, absorbed by the circulation of the splits. These are the basal formulas for splitting, from which any of the factors may be calculated by transposition.

Proportional Division of Air-Current.—Different proportions of air are required in the several splits of a mine than would be obtained by the natural division of the air-current. For example, the longer splits employ a larger number of men and require a larger quantity of air to pass through them. They, moreover, liberate a larger quantity of mine gases, for which they require a larger quantity of air than is passing in the smaller splits. The natural division of the air-current would give to these longer splits less air, and to the shorter ones a larger amount of air, which is directly the reverse of what is needed. On this account, recourse must be had to some means of dividing this air proportionately, as required. This is accomplished by the use of regulators, of which there are two general types, the box regulator and the door regulator.

The *box regulator* is simply an obstruction placed in those airways that would naturally take more air than the amount required. It consists of a brattice or door placed in the entry, and having a small shutter that can be opened a certain amount. The shutter is so arranged as to allow the passage of more or less air, according to the requirements.

The *door regulator* divides the air made at the mouth of the split. It consists of a door hung from a point of the rib between two entries, and swung into the current so as to cut the air like a knife. The door is provided with a set lock, so that it may be secured in any position, to give more air to the one or the other of the splits, as required. The position of this regulator door, as well as the position of the shutter in the box regulator, is always ascertained practically by trial. The door is set so as to divide the area of the airway proportionate to the work absorbed in the respective splits. The pressure in any split is not increased, each split retaining its natural pressure.

Calculation of Pressure for Box Regulators.—When any required division of the air-current is to be obtained by the use of box regulators, these are placed in all the splits, save one. This split is called the *open*, or *free*, *split*, and its pressure is calculated in the usual way by the formula $p=\frac{ksq^2}{a^3}$. The natural pressure in this open split determines the pressure of the entire mine, as all the splits are subject to the same pressure in this form of splitting.

First, determine in which splits regulators will have to be placed, in order to accomplish the required division of the air. Calculate the natural pressure, or pressure due to the circulation of the air-current, for each split, when passing its required amount of air, using the formula $p=\frac{ksq^2}{a^3}$. The split showing the greatest natural pressure is taken as the free split. In each of the other splits, box regulators must be placed, to increase the pressure in those splits; or, in other words, to increase the resistance of those splits per unit of area.

The size of opening in a box regulator is calculated by the formula for determining the flow of air through an orifice in a thin plate under a certain head or pressure. The difference in pressure between the two sides of a box regulator is the pressure establishing the flow through the opening, which corresponds to the head h in the formula $v=\sqrt{2gh}$. This regulator is usually placed at the end of a split or airway, and as the regulator increases the pressure in the lesser split so as to make it equal to the pressure in the other split, the pressure due to the regulator will be equal to the ventilating pressure at the mouth of the split, less the natural pressure or the pressure due to friction in this split. Hence, when the position of the regulator is at the end of the split, the pressure due to friction in the split is first calculated by the formula $p=\frac{ksq^2}{a^3}$, and this pressure is deducted from the ventilating pressure of the free or open split, which gives the pressure due to the regulator. This is then reduced to inches of water gauge, and substituted for i in the

formula $A = \frac{.0004q}{\sqrt{i}}$. The value of A thus obtained is the area, in square feet, of the opening in the regulator.

By the use of the box regulator, the pressure in all the splits is made equal to the greatest natural pressure in any one. This split is made the open or free split, and its natural pressure becomes the pressure for all the splits, or the *mine pressure.* This mine pressure, multiplied by the total quantity of air in circulation (the sum of the quantities passing in the several splits), and divided by 33,000, gives the horsepower upon the air, or the horsepower of the circulation.

Size of Opening for a Door Regulator.—The sectional area at the regulator is divided proportionately to the work to be performed in the respective splits according to the proportion $A_1 : A_2 = u_1 : u_2$. Or as $A_1 + A_2 = a$, $A_1 : a = u_1 : u_1 + u_2$, and $A_1 = \frac{u_1}{u_1 + u_2} \times a$. This furnishes a method of proportionate splitting in which each split is ventilated under its own natural pressure. The same result would be obtained by the placing of the box regulator at the intake of any split, thereby regulating the amount of air passing into that split, but the door regulator presents less resistance to the flow of the air-current. The practical difference between these two forms of regulators is that in the use of the box regulator each split is ventilated under a pressure equal to the natural pressure of the open or free split, which very largely incrcases the horsepower required for the ventilation of the mine; while in the use of the door regulator each split is ventilated under its own natural pressure, and the proportionate division of the air is accomplished without any increase of horsepower.

In the use of the door regulator, each split is ventilated under its own natural pressure, and hence, in the calculation of the horsepower of such a circulation, the power of each split must be calculated separately, and the sum of these several powers will be the entire power of the circulation.

SPLITTING FORMULAS

Primary Splits.—In the accompanying table are given the formulas used in the calculation of primary splits in the natural division. The letters represent the same quantities they did in the ventilating formulas already given.

VENTILATING FORMULAS FOR PRIMARY SPLITS

To Find	No.	Formula
Pressure, potential	35	$X_p = a\sqrt{\frac{a}{ks}}$ $\Sigma X_p = (X_1 + X_2 + \text{etc.})$
Natural division	41	$q = \frac{X_p}{\Sigma X_p} \times Q$
Pressure	42	$p = \frac{Q^2}{(\Sigma X_p)^2}$
Power	43	$u = \frac{Q^3}{(\Sigma X_p)^2}$
Quantity	44 45	$Q = \Sigma X_p \sqrt{p}$ $Q = \sqrt[3]{(\Sigma X_p)^2 u}$
Increase of quantity due to splitting the pressure being constant	46	$Q = \frac{\Sigma X_p}{X_{p-o}} \times q_o$
Increase in quantity due to splitting, the power being constant	47	$Q = q\sqrt[3]{\left(\frac{\Sigma X_p}{X_{p-o}}\right)^2}$

Example.—What is the pressure potential necessary to ventilate a mine that has one primary split 4 ft. × 5 ft. × 800 ft. and one 4 ft. × 5 ft. × 1,200 ft?

SOLUTION.—Substituting in formula 35, the pressure potential of the first split is $20\sqrt{\frac{20}{.0000000217\times 14{,}400}}=5{,}060$; and for the second split, $20\sqrt{\frac{20}{.0000000217\times 21{,}600}}=4{,}131$. Therefore the pressure potential for the mine is $5{,}060+4{,}131=9{,}191$ units.

VENTILATING FORMULA FOR SECONDARY SPLITS

To Find	No.	Formula
Relative potential for pressure	35	$X_p=a\sqrt{\frac{a}{s}}$
Natural division	48	$q_1=Q-q_2$ $q_2=\dfrac{Q}{1+X_1\sqrt{\frac{1}{X_2{}^2}+\frac{1}{(X_3+X_4)^2}}}$
	41	$q=\frac{X_p}{\Sigma X_p}\times Q$
Pressure	49	$p=k\left(\dfrac{Q}{X_1+\dfrac{1}{\sqrt{\frac{1}{X_2{}^2}+\frac{1}{(X_3+X_4)^2}}}}\right)^{2p}$
Power	50	$u=k\dfrac{Q^3}{\left(X_1+\dfrac{1}{\sqrt{\frac{1}{X_2{}^2}+\frac{1}{(X_3+X_4)^2}}}\right)^2}$
Quantity	51	$Q=\left(X_1+\dfrac{1}{\sqrt{\frac{1}{X_2{}^2}+\frac{1}{(X_3+X_4)^2}}}\right)\sqrt{\frac{p}{k}}$

Secondary Splits.—In the calculation of secondary splits in the natural division, the work is often shortened, when many splits are concerned, by using the relative potential, omitting the factor k. But the final result must then be multiplied by k to obtain the pressure or power; or, these factors must be divided by k, when finding the quantity, as in formulas 49 to 51.

EXAMPLE.—How much air, in cubic feet per minute, is required to ventilate each split, in the natural division, having four secondary splits, as follows: one 4 ft. × 5 ft. × 800 ft.; one 4 ft. × 5 ft. × 500 ft.; one, 4 ft. × 5 ft. × 400 ft.; and one 4 ft. × 5 ft. × 300 ft?

SOLUTION.—Substituting in formula 48, the amount of air required in the first split is $10{,}000-5{,}388=4{,}612$ cu. ft.; and in the second split

$$\frac{10{,}000}{1+.7471\sqrt{\dfrac{1}{9{,}428^2}+\dfrac{1}{(1.0541+1.2172)^2}}}=5{,}388 \text{ cu. ft.}$$

The amount of air required in the third split, from formula 41, is

$$\frac{1.0541}{(1.0541+1.2172)}\times 5{,}388=2{,}500 \text{ cu. ft.};$$

and in the fourth split,

$$\frac{1.2172}{(1.0541+1.2172)}\times 5{,}388=2{,}888 \text{ cu. ft.}$$

Proportionate Division of Air.—To accomplish the proportionate division of air in primary splits, the pressure in one split must be increased by means of a regulator to make it equal to the pressure in the free or open split. Hence the pressure due to the regulator is equal to the difference between the natural pressures in these splits. The pressure due to the regulator may, therefore, be found by the formula

$$p=p_2-p_1,$$

in which p = pressure due to regulator;
p_2 = pressure in free split;
p_1 = pressure in regulator split.

The pressure due to friction may be found by means of formula 13 and the area of the opening in the regulator by formula 37.

EXAMPLE.—What is the pressure due to friction if one split is 4 ft. × 5 ft. × 800 ft. and has 3,500 cu. ft. per min. of air passing through it and the other split is 4 ft. × 5 ft. × 1,200 ft. and has 6,500 cu. ft. per min. of air passing through it?

SOLUTION.—Substituting in formula 13, the pressure due to friction in the first split is $3{,}500^2 \div 5{,}060^2 = .47845$ lb.; and in the second split, $6{,}500^2 \div 4{,}131^2 = 2.4757$ lb.

When using the relative potential, in all calculations relating to secondary splits, multiply the result by k to obtain the pressure or the power. To find the pressure due to friction, free split, and secondary pressure, use formula 13; the areas of the openings in the regulators may be found by formula 37.

EXAMPLE.—What is the pressure in each secondary split if 3,500 cu. ft. per min. of air passes through a split 4 ft. × 5 ft. × 800 ft.; 6,500 cu. ft. per min. passes through a split 4 ft. × 5 ft. × 500 ft.; 4,000 cu. ft. per min. passes through a split 4 ft. × 5 ft. × 400 ft.; and 2,500 cu. ft. per min. passes through a split 4 ft. × 5 ft. × 300 ft., the constant k in each case being .0000000217?

SOLUTION.—Applying formula 13 and multiplying by k gives as the pressure

In the first split,

$$.0000000217 \times (3{,}500 \div .7471)^2 = .47848 \text{ lb.}$$

In the second split,

$$.0000000217 \times (6{,}500 \div .9428)^2 = 1.0314 \text{ lb.}$$

In the third split,

$$.0000000217 \times (4{,}000 \div 1.0541)^2 = .31248 \text{ lb.}$$

In the fourth split,

$$.0000000217 \times (2{,}500 \div 1.2172)^2 = .091546 \text{ lb.}$$

As the natural pressure in the third split is greater than that in the fourth; the third is the free split and its natural pressure is the pressure for the secondary splits. The pressure for the primary splits is then found by first adding the pressures in the second and third, and if their sum is greater than the natural pressure for the first, it becomes the pressure for the primary splits, or the mine pressure. If the natural pressure for the first is the greater, this is made the free split, and its natural pressure becomes the primary or mine pressure. In this case,

the secondary pressure must be increased by placing a regulator in the third split.

If the primary or mine pressure is p_2+p_3, the pressure due to the regulators is p_3-p_4, and $(p_2+p_3)-p_1$.

VENTILATING METHODS AND APPLIANCES

There are, in general, three systems of ventilation, with respect to the ventilating motor employed: natural ventilation, furnace ventilation, and mechanical ventilation.

Natural ventilation means such ventilation as is secured by natural means, or without the intervention of artificial appliances, such as the furnace, or any mechanical appliances by which the circulation of air is maintained. In natural ventilation, the ventilating motor or air motor is the air column that exists in the downcast shaft by virtue of the greater weight of the downcast air and forces the air through the airways of the mine. An air column always exists where the intake and return currents of air pass through a certain vertical height, and have different temperatures.

In any ventilation, air columns are always established in slopes and shafts, owing to the relative temperatures of the outside and inside air. The temperature of the upcast, or return column, may always be assumed to be the same as that of the inside air. The temperature of the downcast, or intake column, generally approximates the temperature of the outside air, although, in deep shafts or long slopes, this temperature may change considerably before the bottom of the shaft or slope is reached, and consequently the average temperature of the downcast, or intake, is often different from that of the outside air. The difference of temperatures will also vary with the season of the year. In winter the outside temperature is below that of the mine, and the circulation in shafts and slopes is assisted, as the return columns are warmer and lighter than the intake columns for the same circulation. In the summer season, however, the reverse of this is the case. The course of the air-current will thus often be changed. When the outside temperature approaches the average temperature of the mine, there will be no ventilation, except such as is caused by accidental wind pressure.

In *furnace ventilation* the temperature of the upcast column is increased above that of the downcast column by means of a furnace. The chief points to be considered are the arrangement and size of the furnace. Furnace ventilation should not be applied to gaseous seams, and in some cases is prohibited by law; it is, however, in use in many mines liberating gas. In such cases the furnace fire is fed by a current of air taken directly from the air-course, sufficient to maintain the fire, and the return current from the mines is conducted by means of a dumb drift, or an inclined passageway, into the shaft, at a point from 50 to 100 ft. above the seam. At this point, the heat of the furnace gases is not sufficient for the ignition of the mine gases. The presence of carbon dioxide in the furnace gases also renders the mine gases inexplosive. In other cases where the dumb drift is not used, a sufficient amount of fresh air is allowed to pass into the return current to insure its dilution below the explosive point before it reaches the furnace.

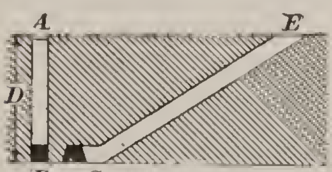


FIG. 1

In a slope opening, the air column is inclined; it is none the less, however, an air column, and must be calculated in the same manner as a vertical column whose vertical height corresponds to the amount of dip of the slope. Fig. 1 shows a vertical shaft and a slope, the air column in each of these being the same for the same temperature. The air column in all dips and rises must be estimated in like manner, by ascertaining the vertical height of the dip.

Calculation of Ventilating Pressure in Furnace Ventilation. The ventilating pressure in the mine airways, in natural or in furnace ventilation, is caused by the difference of the weights of the primary and secondary columns. Air always moves from a point of higher pressure toward a point of lower pressure, and this movement of the air is caused by the difference between these two pressures. In this calculation each column is supposed to have an area of base of 1 sq. ft. Hence, if the weight of 1 cu. ft. of air at a given barometric pressure, and having a temperature equal to the average temperature of the column, is multiplied by the vertical height D of the column,

not only is the weight of the column obtained but the pressure at its base due to its weight. Now, as the ventilating pressure per square foot in the airway is equal to the difference of the weights of the primary and secondary columns,

$$p=\left(\frac{1.3253\times B}{459+t}-\frac{1.3253\times B}{459+T}\right)\times D$$

Calculation of Motive Column or Air Column.—It is often convenient to express the ventilating pressure p pound per square foot in terms of air column or motive column M, in feet. The height of the air column M is equal to the pressure p divided by the weight w of 1 cu. ft. of air, or $M=\frac{p}{w}$. The expression for motive column may be written in terms of the upcast air or of the downcast air, the former giving a higher motive column than the latter for the same pressure, because the upcast air is lighter than the downcast. As the surplus weight of the downcast column of air produces the ventilating pressure, it is preferable to write the air column in terms of the downcast air, or, in other words, to consider the air column as being located in the downcast shaft, and pressing the air downwards, and through the airways of the mine. If the expression for the ventilating pressure is divided by the weight of 1 cu. ft. of downcast air $\left(\frac{1.3253\times B}{459+t}\right)$, after simplifying the motive column, $M=\left(\frac{T-t}{459+T}\right)\times D$, which is the expression for motive column in terms of the downcast air.

If, on the other hand, the expression for the ventilating pressure is divided by the weight of 1 cu. ft. of upcast air $\left(\frac{1.3253\times B}{459+T}\right)$, $M=\left(\frac{T-t}{459+t}\right)\times D$, which is the expression for motive column in terms of the upcast air.

Mechanical Ventilators.—A large number of mechanical ventilators have been invented and applied to the ventilation of mines. Mechanical motors of the fan type present two distinct modes of action in producing an air-current: (1) by propulsion of the air; and (2) by establishing a pressure due to the centrifugal force incident to the revolution of the fan.

Fans have been constructed to act wholly on one or the other of these principles, while others have been constructed to act on both of these principles combined.

The action of the *disk fan* resembles that of a windmill, except that in the latter the wind drives the mill, while in the former the fan propels the air or produces the wind. This type of fan consists of a number of vanes radiating from a central shaft, and inclined to the plane of revolution. The fan is set up in the passageway between the outer air and the mine airways. Power being applied to the shaft, the revolution of the vanes propels the air, and produces a current in the airways. The fan may force the air through, or exhaust the air from, the airways, according to the direction of its revolution. This type of fan is most efficient under light pressures. It has found an extensive application in mining practice, but has been replaced to a large degree in the ventilation of extensive mines; this type of fan acts wholly by propulsion.

Centrifugal fans include all fans that act solely on the centrifugal principle, and those that combine the centrifugal and propulsion principles. The action of the fan depends on the form of the fan blades, which are set at right angles to the plane of revolution, and not inclined, as in the disk fan just described. The blades may, however, be either radial, sometimes spoken of as *paddle blades*, or they may be inclined to the radius either forwards in the direction of revolution, or backwards. When the blades are radial, the action of the fan is centrifugal only. The inclination of the blades backwards from the direction of motion gives rise to an action of propulsion, in addition to the centrifugal action of the fan. The blades in this position may be either straight blades in an inclined position, as in the original Guibal fan, or they may be curved backwards in the form of a spiral, as in the Schiele and Waddle fans.

Centrifugal fans may be exhaust fans or force fans or blowers. In each, the action of the fan is essentially the same; i. e., to create a difference of pressure between its intake or central opening, and its discharge at the circumference. The centrifugal force developed by the revolution of the air between the blades of the fan causes the air within the fan to crowd toward the circumference; as a result, a depression is caused at the

center and a compression at the circumference, giving rise to a difference of pressure between the intake and the discharge of the fan.

Exhaust Fans.—If the intake opening of the fan is placed in connection with the mine airways, and the discharge is open to the atmosphere, the fan will act to create a depression in the fan drift leading to the mine, which will cause a flow of air through the mine airways and into and through the fan. In this case, the fan is exhausting, its position being ahead of the current that it produces in the airway. The atmospheric pressure at the intake of the mine forces the air or propels the current toward the depression in the fan drift caused by the fan's action.

Force Fans and Blowers.—If the discharge opening of the fan is placed in connection with the mine airways, a compression will result in the fan drift owing to the fan's action, and the air will flow from this point of compression through the airways of the mine, and be discharged into the upcast, and thence into the atmosphere. The ventilating pressure in the case of either the exhaust fan or the force fan is equal to the difference of pressure created by the fan's action. In the former case, when the fan is exhausting, the absolute pressure in the fan drift is equal to the atmospheric pressure less the ventilating pressure, while in the latter case, when a fan is forcing, the absolute pressure in the fan drift is equal to the atmospheric pressure increased by the ventilating pressure. This gives rise to two distinct systems of ventilation, known as the vacuum and the plenum system. In the *vacuum system*, the ventilation of the mine is accomplished by creating a depression in the return airway of the mine. In the *plenum system*, the air-current is propelled through the mine airways by means of the compression or ventilating pressure created at the intake opening of the mine.

Position of Fan.—The position of the fan, whether used as an exhaust or blower, should be sufficiently removed from the fan shaft to avoid damage to the fan in case of explosion in the mine. Even in non-gaseous mines, the fan should be located a short distance back from the shaft mouth, to avoid damage due to settlement. Connection should be made with the fan shaft by means of an ample drift, which should be deflected

into the shaft so as to produce as little shock to the current as possible. In cases of gaseous seams, explosion doors should be provided at the shaft mouth. The ventilator in every large mine should be arranged so that it may be converted from an exhaust to a blow-down fan at short notice. This is managed by housing the central orifices or intake of the fan in such a manner as to connect them directly with the fan drift.

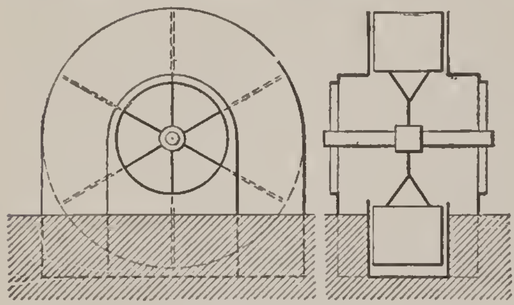


FIG. 2

Types of Centrigufal Fans.—The *Nasmyth fan*, shown in Fig. 2 is the original type of fan. It had straight paddle blades radiating from the center, and was probably the earliest attempt to apply the centrifugal principle to a mine ventilator. Although not recognized at the time, the fan embodied some of the most essential principles in centrifugal ventilation.

About 1850, Biram attempted to improve upon the Nasmyth ventilator by reducing the depth of blade so that it was but one-tenth of the radius. In the Biram *ventilator*, shown in Fig. 3, a large number of straight blades were used but they inclined backwards from the direction of motion at a considerable angle. This fan was run at a considerable speed, but proved very inefficient. It depended more on the effort of propulsion given to the air than on the centrifugal principle, as the depth of the blade was as much too small as that of Nasmyth's was too great. The intake or central opening in this fan was as contracted as in the former type.

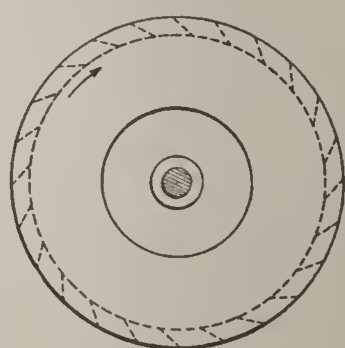


FIG. 3

In the *Waddle ventilator*, Fig. 4, the inventor attempted to reenforce the discharge pressure at the circumference against the pressure of the atmosphere. The discharge took place all

around the entire circumference of the fan, which was entirely opened to the atmosphere. The blades were curved backwards

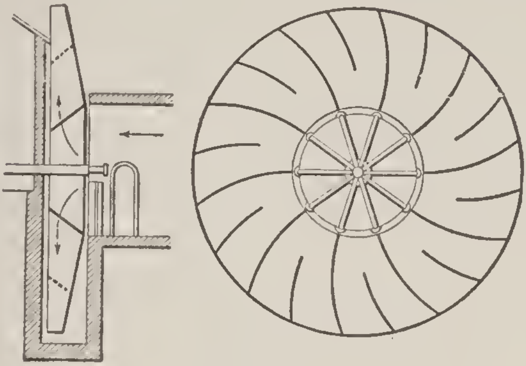


FIG. 4

from the direction of motion in spiral form. The width of the blade decreased from the throat toward the circumference, so as to present an inverse ratio to the length of radius. Thus, the area of passage between the fan blades was maintained constant from the throat to the circumference of the fan.

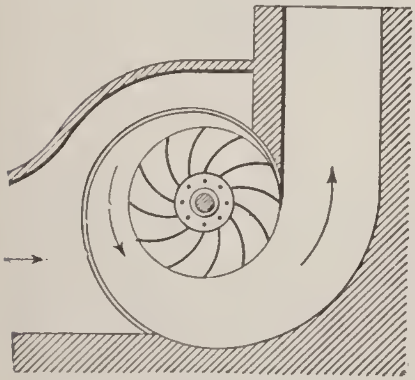


FIG. 5

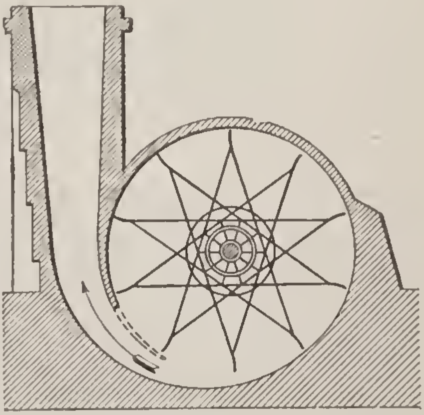


FIG. 6

This is the best type of open-running fan having no peripheral casing, and discharging air into the atmosphere all around the circumference.

The *Schiele ventilator*, shown in Fig. 5, was constructed on the same principles as the Waddle ventilator, but differed from it

by having the discharge made into a spiral chamber surrounding the fan and leading to an expanding chimney.

The *Guibal ventilator*, shown in Fig. 6, embodied the features of the Nasmyth ventilator, with the addition of a casing built over the fan to protect its circumference. This casing was, however, a tight-fitting casing, and as such, differed very materially from the Schiele casing. In the Guibal fan the blades were arranged upon a series of parallel bars passing upon each side of the center and at some distance from it. By this construction, the blades were not radial at their inner edge or the throat of the fan. They were curved, however, as they approached the circumference of the fan, so as to be normal or radial at the circumference.

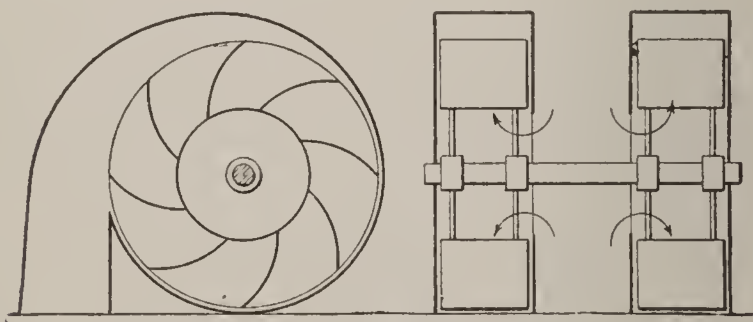


FIG. 7

The *Murphy ventilator*, shown in Fig. 7, consists of twin fans supported on the same shaft and set a few feet apart. Each fan receives its air on one side only, the openings being turned toward each other. This ventilator is built with a small diameter, and is run at a high speed. The blades are curved backwards from the direction of motion. The intake opening is considerably enlarged; a spiral casing generally surrounds the fan, and in every respect this fan makes an efficient high-speed motor. It has received considerable favor in the United States, where it has been introduced into a large number of mines.

Perhaps no centrifugal ventilator has been as little understood in regard to its principle of action as the *Capell fan*, shown in

Fig. 8. The fan is constructed along the lines of the Schiele ventilator, but differs from that ventilator in the manner of receiving its intake air and delivering the same into the main body of the fan. A set of smaller supernumerary blades occupy a cylindrical space within the main body of the fan, and are inclined to the plane of revolution so as to assist in deflecting the entering air through small ports or openings into the main body of the fan, where it is revolved and discharged at the circumference into a spiral space resembling that surrounding

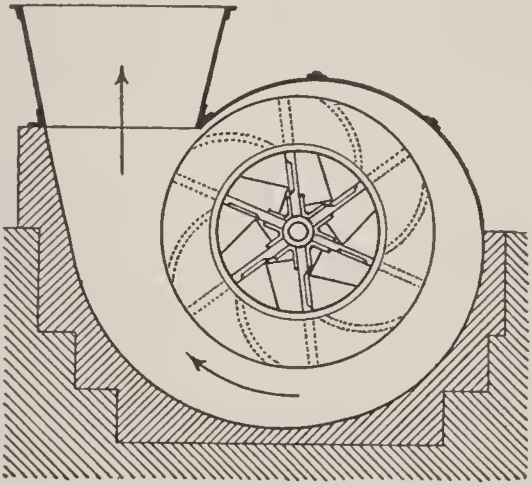


Fig. 8

the Schiele fan. The larger blades of this fan are curved backwards as the Schiele blades, but are not tapered toward the circumference. The fan is capable of giving a high water gauge, and is efficient as a mine ventilator. The space surrounding the fan is extended to form an expanding chimney. The fan may be used either as an exhaust fan or a blower. The best results in the United States have been obtained by blowers; in Germany, where this fan is in general use, there are no blowers.

Conducting Air-Currents.—A *mine door* is used for the purpose of deflecting the air-current from its course in one entry so as to cause it to traverse another entry, at the same time permitting the passage of mine cars through the first entry.

Stoppings are used to close break-throughs that have been made through two entries, or rooms, for the purpose of maintaining the circulation as the workings advance; also to close or seal off abandoned rooms or working places. Stoppings must be air-tight and substantially built.

An *air bridge* is constructed for the passage of air across another airway. When it crosses over, it is called an *overcast;* when it passes under the airway, it is called an *undercast.* In almost every instance, overcasts are preferable to undercasts.

An *air brattice* is any partition erected in an airway for the purpose of deflecting the current. A thin board stopping is sometimes spoken of as a brattice; but the term applies more particularly to a thin board or canvas partition running the length of an entry or room and dividing it into two airways, so that the air will pass up one side of the partition and return on the other side, thus sweeping the face of the heading or chamber. Such a temporary brattice is often constructed by nailing cloth to upright posts set from 4 to 6 ft. apart along one side of the entry a short distance from the rib.

Curtains are sometimes called *canvas doors;* they are heavy duck, or canvas, hung from the roof of the entry to divide the air or deflect a portion of it into another chamber or entry. Curtains are thus used very often previous to setting a permanent door frame. They are of much use in longwall work, or where there is a continued settlement of the roof, which would prevent the construction of a permanent door; also, in temporary openings where a door is not required.

HOISTING AND HAULAGE

HOISTING

There are two general systems of hoisting in use: (1) Hoisting without attempting to balance the load, when the cage and its load are hoisted by an engine and lowered by gravity; (2) Hoisting in balance, when the descending cage or a special counterbalance assists the engine to hoist the loaded ascending cage. Hoisting in balance is usually effected by the use of

double cylindrical drums, flat ropes winding on reels, conical drums, the Koepe system, and the Whiting system.

Double cylindrical drums are widely used. They consist essentially of an engine coupled directly or else geared to the common axis of the drums. The drums are usually provided with friction or positive clutches, and brakes, so that they can be run singly if desired, or the load can be lowered by gravity and the brake.

Flat ropes wound on reels are sometimes used either for unbalanced hoisting with a single reel or for balanced hoisting with a double reel. A flat rope has the advantage of preventing fleeting, but its first cost, extra weight, wear, and difficulty of repairing have prevented its very general adoption.

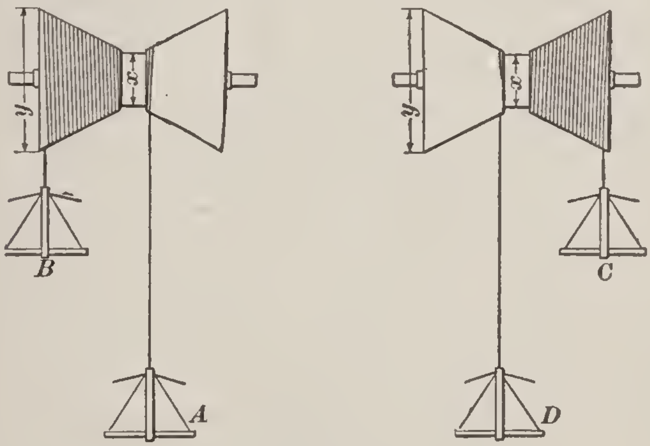


FIG. 1

A *conical drum*, Fig. 1, equalizes the load on an engine just as a flat rope on a reel. On account of the fleeting of the rope, however, the drum must be set at a considerable distance from the shaft to prevent the rope leaving the head-sheave. A tail-rope gives the most perfect counterbalance, the weight of the cage and rope on each side being exactly equal.

In the *Koepe system*, Fig. 2, one rope runs over and the other under driving sheaves S. A tail-rope R is used, and the head-sheaves x, x' are placed vertically and at such an angle to each other that their grooves and the groove in the driving sheave are in line. As the main driving shaft is short, the engines can

be placed close together, thus requiring a smaller foundation and engine house than for a drum hoist. The objection to the system is the liability of the rope to slipping about the driving sheave, for which reason a hoisting indicator cannot be depended on. The system is also inconvenient for hoisting from different levels in the same shaft; besides should the rope break, both cages will fall to the bottom.

The *Whiting system*, Fig. 3, uses two narrow-grooved drums placed tandem instead of a single-driving sheave as in the Koepe system. The rope passes from the cage *A* over a head-sheave, under the guide sheave *T* and around the sheaves *M* and *F* three times, then out to the fleet sheave *C*, back under another guide sheave, and up over another head-sheave to the cage *B*. The sheave *M* is driven by a motor either coupled direct to its shaft, or geared. The drums *F* and *M* are coupled together by a pair of connecting-rods like the drivers of a locomotive, so that it is possible to utilize all the friction of both drums to drive the rope. A tail-rope is not depended on to produce more friction, though one is generally used as a balance to the loads.

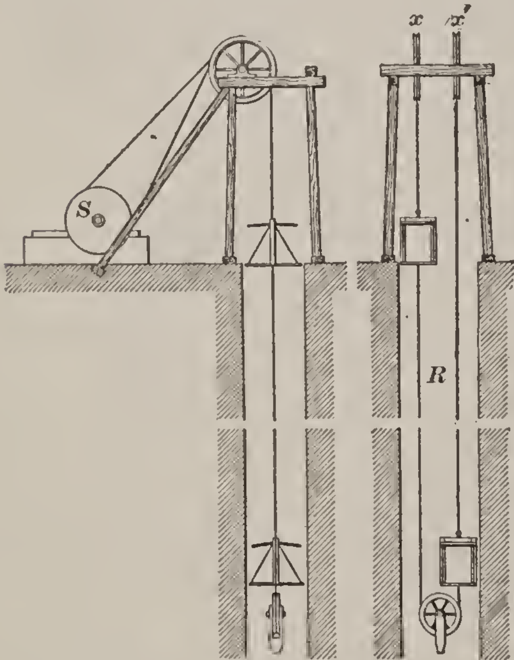


FIG. 2

It is best to incline the follower sheave *F* from the vertical an amount equal in its diameter to the distance between the centers of two adjacent grooves, the object being to eliminate chafing between the ropes around the drums and to prevent them from running off by enabling the rope to run from each groove in one drum straight to the proper groove in the other. This throws

the shaft and crankpins out of parallel with those of the main drum, but this difficulty is overcome by the connections in the ends of the parallel rods. The fleet sheave *C* is arranged to travel backwards and forwards, as shown by the dotted lines, in order to change the working length of the rope, whereby hoisting can be done from different levels in the shaft.

Power Used for Hoisting.—The power used for hoisting is generally steam for the main hoists. Electricity is, however, coming rapidly into use, particularly for smaller hoists and underground installations, and for main hoists in locations

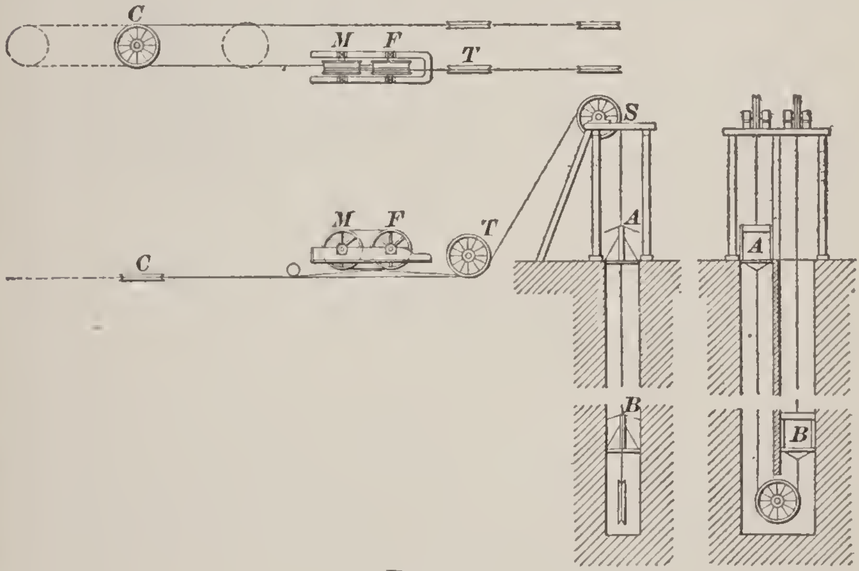


FIG. 3

where fuel is expensive and water-power available. Gasoline engines are also being used to an increasing degree, particularly for smaller hoists and in local installations, and they are said to give very satisfactory results.

Problems in Hoisting.—The most common problems in hoisting are the following:

To Balance Conical Drum.—Having given the diameter of one end of a conical drum, to determine the diameter of the other end that will equalize the load on the engines, call total load at bottom *A*, Fig. 1, empty cage at top *B*, loaded cage at

top C, empty cage plus rope at bottom D, small diameter of drum x, and large diameter y; then,

$$Ax - By = Cy - Dx$$

To Find the Size of Hoisting Engine.—Let

D = diameter of cylinder;
P = mean effective steam pressure in cylinders;
r = ratio of stroke to diameter of cylinder;
w = work per revolution required to be done;

then, by making one cylinder capable of doing the work, n = number of strokes, u = work per minute in foot-pounds.

$$D = 1.97\sqrt[3]{\frac{w}{Pr}}, \text{ or } D = \sqrt[3]{\frac{u}{.7854Prn}}$$

To Find Actual Horsepower of Engine for Hoisting Any Load Out of Shaft at Given Rate of Speed.—To the weight of the loaded car add the weight of the rope and cage. This will give the gross weight.

Then,

$$\text{H. P.} = \frac{\text{gross weight in lb.} \times \text{speed in ft. per min.}}{33{,}000}$$

Add $\frac{1}{2}$ for contingencies, friction, etc.

The following rules regarding winding engines are given by Percy:

To Find Load That Given Pair of Direct-Acting Engines Will Start.—Multiply the area of one cylinder by the average pressure of the steam per square inch in the cylinder, and twice the length of the stroke. Divide this by the circumference of the drum, and deduct $\frac{1}{3}$ for friction, etc.

Knowing Load and Diameter of Cylindrical Drum, and Length of Stroke, Cut-off and Pressure of Steam at Steam Gauge, to Find Area and Diameter of Cylinders of Pair of Direct-Acting Engines. Multiply the load by the circumference of the drum, and add $\frac{1}{2}$ for friction, etc. Divide this by the mean average steam pressure, multiplied by twice the length of the stroke.

To Find Approximate Period of Winding on a Cylindrical Drum With Pair of Direct-Acting Engines.—Assume the piston to travel at an average velocity of 400 ft. per min., and divide this by twice the length of the stroke, and multiply by the circumference of the drum. This gives the speed of cage in feet

per minute. Divide the depth of shaft by this, and the result will be the period of winding.

Head-Frames.—Head-frames are built of wood or steel. They vary in height from 30 to 100 ft., depending on local conditions. The inclined leg of a head-frame should be placed so as to take up the resultant strain due to the load hanging down the shaft and the pull of an engine.

Fig. 4 shows the graphic method of determining the direction and magnitude of this resultant force. Produce the direction of the two portions of the rope leading to the drum and down the shaft until they intersect at *G*, measure off a distance *GK* to scale to represent the load hanging down the shaft; similarly, measure off *GH* to the same scale to represent the pull of the engine, complete the parallelogram *GHLK*; the direction of the line *GL* represents the direction of the resultant force, and its length represents the amount of this force. The inclined leg of the head-frame should be placed as nearly as possible parallel to this resultant line, and should be designed to withstand a compressive strain equal to this resultant.

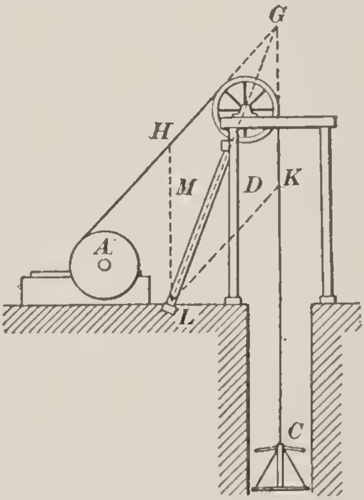


FIG. 4

Head-sheaves are made of iron, being sometimes entirely cast, or else the rim and hub are cast separately and wrought-iron spokes are used. The former are cheaper and quite satisfactory, but the latter are lighter and stronger, and therefore usually better. The diameter of the sheave depends on the diameter of the rope, and the table giving this will be found on page 106. The groove in the sheave should be wood-lined, to reduce wear on the rope. Wrought-iron spokes should be staggered in the hub and not placed radially.

Safety catches usually consist of a pair of toothed cams placed on either side of the cages and enclosing the guides. When the load is on the hoisting rope, these cams are kept away from the guides by suitable springs; but if the rope breaks, the springs

come into action and throw the catches or dogs so that they grip the guides, then the tendency to fall increases the grip on the guides.

Detaching hooks are devices that automatically disconnect the rope from the cage in case of overwinding.

HAULAGE

The magnitude of modern mines and the practice of loading or of treating the coal or ore at a large central station makes the underground haulage of the material one of the most important problems in connection with mining. A good haulage system is now essential to make most mines a commercial success.

Gravity Planes.—With gravity planes the loaded car or trip hauls the empty car up the grade. Two ropes are fastened to a drum so that the rope attached to the loaded car unwinds from the drum as the car descends, while the rope secured to the empty car is wound on the drum and the car thus hauled up the plane. The following rule gives suggestions based on practice that have been successful: For lengths not exceeding 500 ft., the minimum grade for the incline should be 5% when the weight of the descending load is 8,000 lb. and that of the ascending load 2,800 lb. Or the inclination should not be less than 5½% if the respective descending and ascending loads are one-half of those just given. When the length of the-plane is from 500 to 2,000 ft., the grade should be increased from 5% to 10%, according to the loads. A load of 4,000 lb. on a 10% grade 2,000 ft. long will hoist a weight of 1,400 lb.

The *angle of inertia* is that angle or inclination at which a car will start to move down the slope or plane. The car, when it has once started on this grade, will continue to accelerate its speed as it descends the plane AB, in the accompanying illustration. If the angle of inclination is decreased until the plane AB occupies the position AC, so that the moving car will continue to move at a uniform velocity instead of accelerating its speed, the angle DCA will be the *angle of rolling friction*, and the tangent of this angle will be the *coefficient of rolling friction* for the car.

The upper portion of a plane is made steeper than the lower portion so that the trip may start quickly at the head and afterwards maintain a uniform velocity. With a good brake to control the cars, the uniform grade of a central portion of a gravity plane should not fall much below 3°, which corresponds practically to a $5\frac{1}{4}\%$ grade.

Rope Haulage.—The *tail-rope system* of haulage uses two ropes and a pair of drums on the same shaft. The main rope passes from one drum directly to the front of the loaded trip, and the tail-rope passes from the other drum to the large sheave wheel at the end of the road, and back to the rear of the loaded trip. While hauling the loaded trip, the drum on which the tail-rope is wound is allowed to turn freely on its journal by throwing its clutch out, while the engine turns the other drum. When the empty trip is being hauled, the clutch on the main-rope drum is thrown out and the one on the tail-rope drum is thrown in. The engine then turns the tail-rope drum and allows the other one to pay out rope as the trip advances.

The tail-rope system is suitable for steep, circuitous, and undulating roads. The trip can be kept stretched at all points, and thus the cars will be prevented from bumping together or from being jerked apart as the trip is passing over changes in the grade. It is undoubtedly the most satisfactory system of rope haulage under the natural conditions of most haulage roads in mines, and especially so where but one road is available for haulage purposes.

Calculation of Tension of Haulage Rope.—The tension of haulage ropes may be found by the formula,

$$T = W(\sin a + \mu \cos a) + w(d + \mu l),$$

in which T = tension or pull upon rope, in pounds;

W = weight of loaded trip, in pounds;

w = weight of rope per linear foot, in pounds;

l = length of two ropes; equals 2 times the distance from winding drum to tail-sheave, in feet;

d = vertical drop of rope, in feet;

a = slope angle of maximum grade.

Example.—What size of steel wire rope will be required to haul a trip of 20 mine cars, the weight of the loaded cars being 3,000 lb. each, the depth of the shaft 300 ft., and the distance

from the foot of the shaft to the tail-sheave 900 yd., the maximum grade in this haulage being 10°, $\mu = \frac{1}{40}$? Assuming a $\frac{3}{4}$ in. rope, weighing .89 lb. per lin. ft.

SOLUTION.—From the formula just given,

$$T = 60{,}000 \times \left(.17365 + \frac{.9848}{40}\right) + .89 \times \left(300 + \frac{6{,}000}{40}\right)$$

= say 12,300 lb., or somewhat over 6 T.

Referring to the tables for steel haulage ropes with 6 strands of 7 wires each, the breaking strain of a $\frac{3}{4}$ in. rope, weighing .89 lb. per lin. ft., is found to be 18.6 T. which will give a factor of safety of about 3. However, a $\frac{7}{8}$-in. or even a 1-in. rope, should be used for a change of ropes would then be required less often. Making the necessary corrections for 1-in. rope weighing 1.58 lb. per lin. ft., $T = 12{,}607$ lb.

The *endless-rope system* uses an endless rope, which is kept running continuously by a pair of drums geared together and set tandem. The drums are comparatively narrow and provided with grooves for the rope to run in. Two drums are necessary to get sufficient friction to drive the rope when the trip is attached to it. The rope is passed around both drums a number of times, depending on the amount of friction desired, without completely encircling either. It then passes to a tension wheel at the rear of the drums and thence to the sheave wheel at the far end of the road and back to the drums. To be used to best advantage, the grade should be in one direction and it should be necessary to haul cars from a number of places en route. The cars are attached to the rope by friction grips in a manner quite similar to the way in which street cars are attached to cable lines. Therefore, any jerking due to the cars bumping together or stretching the hitchings will seriously injure the rope where the grip takes hold. A double road is an essential feature of endless-rope haulage.

To Determine Friction Pull on an Endless-Rope Haulage.

Let O = output in pounds per minute;
v = speed of winding, in feet per minute;
l = length of haulage road, in feet;
c = capacity of mine car, in pounds;
w_1 = weight of mine car, in pounds;
w = weight of rope, in pounds;

T = load on the rope, in pounds;
μ = coefficient of friction.

Then, $\dfrac{lO}{v}$ = weight of material in transit;

$2\left(\dfrac{lO}{vc}\right)w_1$ = weight of moving cars, loaded and empty;

$2lw$ = weight of rope;

$\dfrac{lO}{v}\left(1+\dfrac{2w_1}{c}\right)+2wl$ = entire moving load.

And if the coefficient of friction equals $\frac{1}{40}$,

$$T=\mu\left[\frac{lO}{v}\left(1+\frac{2w_1}{c}\right)+2wl\right]$$

$$\text{H. P.}=\frac{\mu l}{33{,}000}\times\left[O\left(1+\frac{2w_1}{c}\right)+2wv\right]$$

Inclined Roads.—The calculation of power for inclined roads is the same as that just given, except that the work due to lifting the coal through a height h must be added to that found by the previous formulas. If h equals the elevation due to the grade of the incline, the additional work of the engine due to hoisting the load from this elevation will be Oh and the total work per minute u will be

$$u=\mu l\left[O\left(1+\frac{2w_1}{c}\right)+2wv\right]+Oh$$

Motor Haulage.—Wire-rope haulage is very efficient in headings, on heavy grades, and against large loads, but in crooked passages it entails great costs for renewals and repairs. When the grades do not exceed 5% for short distances and average 3% against, or for short distances 8% and 5% average in favor of loads, locomotives have been found the most economical form of haulage. *Gathering locomotives* are used to take the cars from the rooms. They are similar in their general construction to the ordinary traction locomotive but are shorter and lower.

In general, it costs from 6 to 10c. per T. to deliver coal from face of workings to shaft, slope, or tipple, where the haul is 1 mi. and the tracks approximately level; yet there are mines that at present haul from parting with the trolley system, the

miner delivering from face of room, making an average round trip of 9,000 ft., at a total cost of 1c. per T. Since the advent of the *electric-mining locomotive*, there has been a change in the mine wagons universally used. Formerly it was customary to find as much as 60 lb. per ton car resistance on the level, while at present it is as low as 15 lb.

Compressed air locomotives are particularly useful in gaseous mines, as they improve ventilation and are perfectly safe under all conditions. Their great disadvantage is their size.

For a number of years *gasoline motors* have been used for various purposes on the Pacific coast and in metal mines of the west, this development being brought about by the high cost of steam generation and, in many cases, scarcity of water in arid regions. In shape and appearance the gasoline locomotive resembles the electric locomotive. They are usually constructed to run on full and on half speed. Each motor is equipped with a carbureter, which properly mixes the air and the gasoline in the cylinder. Each locomotive is also equipped with an electric igniting device, which operates from a storage battery when the motor is starting and thereafter from a magneto. In some motors, absorption chambers are used to absorb the carbon dioxide generated and to cool the gases. These chambers are also a protection against the ignition of gas or coal dust when the engine back fires.

The advantages of the gasoline motor are: No power plant is needed to operate them, the power-generating apparatus being a part of the motor. No transmission-wire lines or pipe lines are needed. Humidification of the air is aided. The disadvantages are: The use of gasoline in mines, as a mixture of gasoline and air forms an explosive gas. Combustion of gasoline extracts oxygen from the air. Carbon dioxide and nitrogen are products of combustion, and if combustion is not complete carbon monoxide is formed. The gasoline motor costs 25 to 50% more than an electric motor of the same power. The gasoline motor will not start as large a trip or take an overload like an electric motor does.

The Bituminous Mine Laws of Pennsylvania say, "No product of petroleum or alcohol or any compound that in the opinion of the inspector will contaminate the air to such an

extent as to be injurious to the health of the miner, shall be used as motive power in any mine." Modern gasoline motors are eliminating this objection more and more as improvements are made.

Speed of haulage depends on the system of haulage used and on the condition of the haulage road. The law in Pennsylvania provides for a speed of haulage not over 6 mi. per hr., and this is the speed at which electric and compressed-air haulages are usually calculated and at which loaded trips are usually run. Empty trips are usually run at a slightly higher speed. It has been found in general practice that the maximum pulling power of a mule as well as a locomotive is, approximately, one-fifth of its weight, or, in other words, a locomotive will pull as much as the same weight of mules will pull, and at a speed about three times as great.

MINE ROADS AND TRACKS

Underground or mine-car tracks should be solidly laid on good sills, resting on the solid floor of the mine. They should be well ballasted, and should have good clean gutters on the lower side of the entry, so that the rails may be protected as much as possible from the action of the mine water.

The *grades* depend entirely on circumstances, but, when possible, the grade should be in favor of the load, and should be at least 5 in. in 100 ft. to insure flow in the gutters alongside the track.

Ties should be spaced about 2 ft. apart, center to center, making 15 to a 30-ft. rail. The rail should be well spiked to the ties with four spikes to each tie, the joint between two rails on one side of the track being located about midway between two joints on the opposite rail. On curves, ties should be laid so as to form radii of the curves of the track.

The weight of *rail* to be chosen in any individual case depends entirely on the weight of cars used, and the motive power. For cars having a capacity of about 1½ T., the weight of rail, when the motive power is live stock, should not be less than 16 lb. per yd., while for cars having a capacity of 2 T. or over, a 20-lb. rail should be used. There is no economy in using a very light rail, as the base is gradually eaten away by the mine water;

a heavy section of rail can be used much longer before the rails become weakened. On main roads, where haulage machinery of one kind or another is used, the weight of rail for 2-T. cars should be from 25 lb. to 35 lb. per yd., and on steep slopes as high as 40 lb. per yd. In the case of locomotive haulage, authorities claim that the weight of rail should be regulated by allowing 1 T. for each driver for each 10 lb. weight of rail per yd.

The *gauge* of the track in coal mines should not be less than 30 in. nor more than 48 in. A mean between these two, or a gauge of from 38 in. to 42 in. is desirable, because it combines, to a certain extent, the advantages claimed for the extremes.

Curves should be of as large a radius as permissible, and never, if possible, of less radius than 25 ft. The resistance of curves is considerable; and the smaller the radius of the curve, and the greater the length of the curved track occupied by the trip, or train, the greater is the resistance.

To Bend Rails to Proper Arc for Any Radius.—Rails are usually 30 ft. long, and the most convenient chord to use in bending mine rails is 10 ft. Then, having the radius and chord, find the rise of middle ordinate by squaring the radius, and from it take the square of one-half the chord. Extract the square root of the remainder and subtract it from the radius; the result will be the rise of the middle ordinate. Thus, having a radius of 30 ft. and a chord of 10 ft. the middle ordinate will be,

$$30 - \sqrt{30^2 - 5^2} = .42 \text{ ft.}$$

Rail Elevation.—In elevating rails on curves, consider whether the hauling is to be done by a rope, a locomotive, or an electric motor. For either of the latter, elevate the rail on the outside of the curve; but for the first, elevate the inner rail, for as the power is applied by a long flexible rope, there is always a tendency for both rope and cars to take the long chord of the curve as soon as the point of curve is reached. On slope haulages, operated by a single rope, when the weight of the cars traveling on the grade of the slope is sufficient to draw the rope off the hoisting drum, the rails on curves should be elevated on the outside, the effect then being similar to that of a locomotive, i. e., the centrifugal force tends to throw the car to the outside

of the track. In such cases, the elevation should be moderate so as not to interfere with the trip when drawn out again by the rope—the opposite effect being then experienced.

Rollers.—The rollers on level tracks should not be more than about 20 ft. apart to properly carry the rope, and on gravity slopes where the lower end of the slope gradually flattens off, the distance between rollers should not be more than 12 to 15 ft., as this spacing allows the trip of cars to run much farther by keeping the rope well off the ties, than if they are farther apart, thereby not supporting the rope, and causing a great amount of friction between the rope and the ties.

Switches.—The switch, or *latch*, most commonly used in mines is shown in Fig. 1. When the branch or siding is in constant use, an ordinary railway frog is substituted for the cross-bar *b*. The latches *a*, are wedge-shaped bars of iron (made as high as the rail) with an eye in the thick end.

A modification of this switch is shown in Fig. 2, which represents a form of double switch. These latches are set by the drivers, who kick them over and drop a small square of plate iron between them to hold them in place. This switch costs more than the other style and is better adapted to outside roads than to inside roads. The ordinary movable rail switch in every-day use on all surface railways is sometimes used in mine roads. It is commonly used in slopes arranged as shown by Fig. 6, to replace latches set by the car, and is also largely used in outside roads.

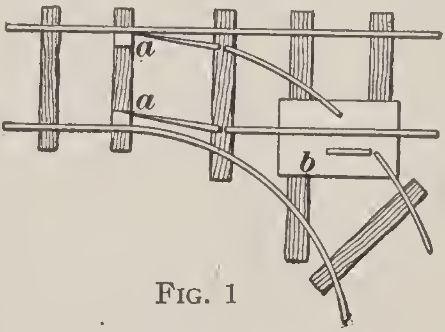


Fig. 1

For crossings, ordinary railway frogs and grade crossings are sometimes used, as is also a small turntable, which then answers two purposes. More frequently the plan shown in Fig. 3, in which four movable bars are thrown across the main track whenever the other road is to be used, is adopted. The subordinate road is built from $1\frac{1}{2}$ to 2 in. higher than the main road, to allow the bars to clear the main-track rails.

Turnouts.—On gangways or headings used as main haulage roads, turnouts should be constructed at convenient

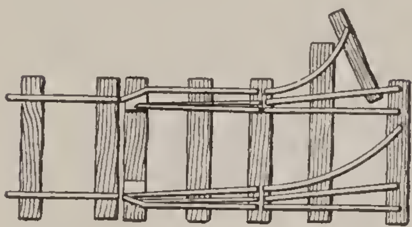


FIG. 2

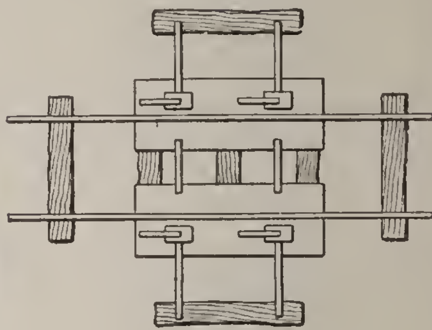


FIG. 3

intervals to allow the loaded and empty trips to pass. These turnouts should be long enough to accommodate from 5 or 6 up to 15 or 20 cars.

Slope Bottoms.—At the foot of a slope or at the landing on any lift, the gangway is widened to accommodate at least two tracks—one for the empty and one for the loaded cars. The empty track should be on the upper side of the gangway, or that side nearer the floor of the seam, and the loaded track on that side of the gangway nearer the roof of the seam. An arrangement of tracks often used is shown in Figs. 4 to 9.

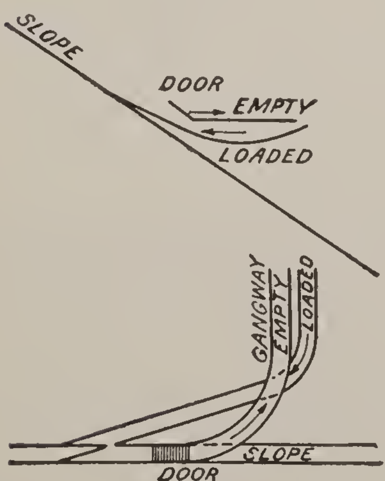


FIG. 4

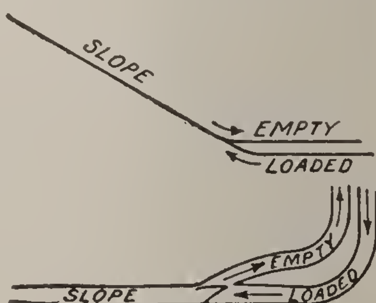


FIG. 5

Material Required for 1,000 Ft. and for 1 Mi. of Single Track.—For the quantity of wooden ties required for 1,000 ft.

and for 1 mi. of single track, see page 307. Weights of rails are given in long tons of 2,240 lb.; hence .9 T. is equal to

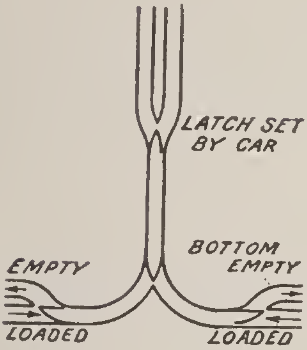


FIG. 6

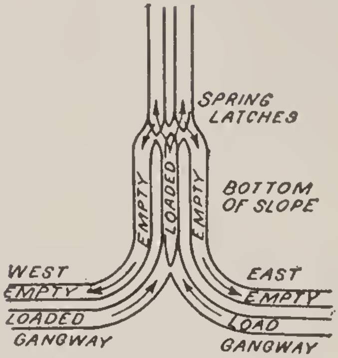


FIG. 7

2,016 lb. and not to 1,800 lb. Each increase of 5 lb. per yd. in the weight of the rail, increases by 1.488 T. and by 7.857 T., respectively, the tons required to lay 1,000 ft. or 1 mi. of track. In measuring a rail, it will be found that the height of a rail is equal to the width of its base.

Rule.—*To find the weight, in long tons, of the rails necessary to lay 1 mi. of single track, multiply the weight per yard of the rail by* $\frac{11}{7}$, *or by* 1.5714.

Rule.—*To find the weight of rails for 1,000 ft. of single track, multiply the weight per yard by* .29761.

Thus, the weight of 70-lb. steel for 1 mi. and for 1,000 ft. of single track would be, respectively, $70 \times \frac{11}{7} = 110$ T., and $70 \times .29761 = 20.833$ T.

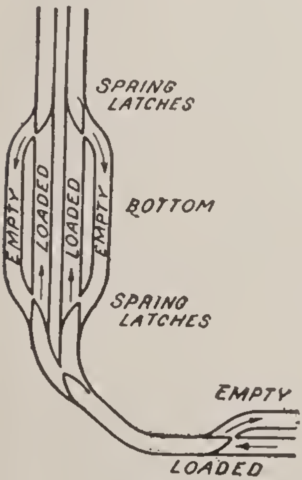


FIG. 8

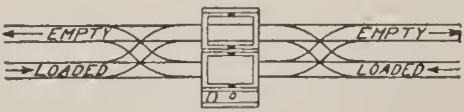


FIG. 9

For lengths other than 1,000 ft., multiply the quantities for 1,000 ft. by the ratio the given length of track bears to 1,000.

Thus for the material for 600 ft., 1,580 ft. or 4,000 ft., multiply the quantities (rails, fish-plates, bolts, or spikes) by .6, 1.58 or by 4 as may be.

Prices quoted for rails include the necessary splice bars (fish-plates) and bolts, but not the spikes. If requested at the time of placing the order, the mills will drill the holes necessary for electric bonding, and, generally, without charge. While the

STANDARD SIZE OF RAILS

Weight per Yard Pounds	Height Inches	Width of Head Inches	Amount Required per 1,000 ft. Tons	Amount Required per Mile Tons
8	$1\frac{1}{2}$	$\frac{13}{16}$	2.381	12.571
12	$1\frac{7}{8}$	$1\frac{1}{16}$	3.571	18.857
16	$2\frac{1}{4}$	$1\frac{1}{4}$	4.762	25.144
20	$2\frac{1}{2}$	$1\frac{3}{8}$	5.952	31.429
25	$2\frac{3}{4}$	$1\frac{1}{2}$	7.441	39.286
30	3	$1\frac{5}{8}$	8.929	47.143
35	$3\frac{1}{4}$	$1\frac{3}{4}$	10.417	55.000
40	$3\frac{1}{2}$	$1\frac{7}{8}$	11.905	62.857
45	$3\frac{11}{16}$	2	13.393	70.714
50	$3\frac{7}{8}$	$2\frac{1}{8}$	14.881	78.571
55	$4\frac{1}{16}$	$\frac{1}{4}$	16.369	86.428
60	$4\frac{1}{4}$	$2\frac{3}{8}$	17.858	94.286
65	$4\frac{7}{16}$	$2\frac{13}{32}$	19.346	102.143
70	$4\frac{5}{8}$	$2\frac{7}{16}$	20.833	110.000
75	$4\frac{13}{16}$	$2\frac{15}{32}$	22.321	117.857
80	5	$2\frac{1}{2}$	23.809	125.714
85	$5\frac{3}{16}$	$2\frac{9}{16}$	25.298	133.571
90	$5\frac{3}{8}$	$2\frac{5}{8}$	26.786	141.429
95	$5\frac{9}{16}$	$2\frac{11}{16}$	28.274	149.286
100	$5\frac{3}{4}$	$2\frac{3}{4}$	29.763	157.143

standard length of rails is 30 ft., the order is considered to have been acceptably filled, if not to exceed 10% of the rails are in shorter lengths, varying by even feet down to 24 ft. In the accompanying table all sizes from 40 lb. to 100 lb., are of the standard established by the American Society of Civil Engineers. A certain quantity of these standard sizes are usually in stock, insuring the prompt filling of small orders.

MATERIALS REQUIRED FOR SINGLE-TRACK ROAD

Size of Road	Number of 30-Ft. Rails Required	Number of Splices Required	Number of Bolts Required	
			4 per Joint	6 per Joint
1,000 ft.......	68	236	272	408
1 mi..........	352	704	1,408	2,112

SIZES AND QUANTITIES OF SPIKES REQUIRED FOR TIES 2 FT., CENTER TO CENTER, 4 SPIKES PER TIE

Size Measured Under Head	Average Number per Keg of 200 Lb.	Quantity Required				Weight of Rail Used
		per 1,000 Ft. Track		per Mile of Track		
Inches		Lb.	Kegs	Lb.	Kegs	Pounds
$2\frac{1}{2}\times\frac{3}{8}$	1,342	300	$1\frac{1}{2}$	1,575	$7\frac{7}{8}$	8 to 16
$3\times\frac{3}{8}$	1,240	324	$1\frac{2}{3}$	1,710	$8\frac{1}{2}$	16 to 20
$3\frac{1}{2}\times\frac{3}{8}$	1,190	340	$1\frac{2}{3}$	1,780	9	16 to 20
$4\times\frac{3}{8}$	1,000	360	$1\frac{4}{5}$	2,090	$10\frac{1}{2}$	16 to 25
$3\frac{1}{2}\times\frac{7}{16}$	900	445	$2\frac{1}{4}$	2,350	11	16 to 25
$4\times\frac{7}{16}$	720	550	$2\frac{3}{4}$	2,910	$14\frac{3}{4}$	20 to 30
$4\frac{1}{2}\times\frac{7}{16}$	680	590	3	3,110	$15\frac{1}{2}$	20 to 30
$4\times\frac{1}{2}$	600	670	$3\frac{1}{3}$	3,520	$17\frac{3}{8}$	25 to 35
$4\frac{1}{2}\times\frac{1}{2}$	530	750	$3\frac{3}{4}$	3,960	20	30 to 35
$5\times\frac{1}{2}$	450	880	$4\frac{2}{5}$	4,660	$23\frac{1}{3}$	35 to 40
$5\times\frac{9}{16}$	400	980	5	5,170	26	40 to 55
$5\frac{1}{2}\times\frac{9}{16}$	375	1,112	$5\frac{3}{5}$	5,870	$29\frac{1}{3}$	45 to 75
$5\frac{1}{2}\times\frac{5}{8}$	300	1,334	$6\frac{2}{3}$	7,040	$35\frac{1}{5}$	75 to 100

NOTE.—When ordering spikes, a reasonable allowance should be made for waste. For ordinary mine track with 2 spikes to the tie, divide the quantities given in the table by 2. For other spacing than 2 ft., proceed as follows: For 30 in., multiply the quantity of spikes by .8; for 28 in., by .858; for 26 in., by .893; for 22 in., by 1.092; for 20 in., by 1.2; and for 18 in., by 1.334.

NUMBER OF BOARD FEET IN MINE TIES

Area of Cross-Section	Length of Tie							
	4 Ft. 6 In	5 Ft.	5 Ft. 6 In.	6 Ft.	6 Ft. 6 In.	7 Ft.	7 Ft. 6 In.	8 Ft.
Inches	Number of Board Feet							
3× 5	5.6250	6.2500	6.8750	7.5000	8.1250	8.7500	9.3750	10.0000
4× 5	7.5000	8.3333	9.1667	10.0000	10.8333	11.6667	12.5000	13.8333
5× 5	9.3748	10.4167	11.4583	12.5000	13.5416	14.5833	15.6250	16.6667
4× 6	9.0000	10.0000	11.0000	12.0000	13.0000	14.0000	15.0000	16.0000
5× 6	11.2500	12.5000	13.7500	15.0000	16.2500	17.5000	18.7500	20.0000
6× 6	13.5000	15.0000	16.5000	18.0000	19.5000	21.0000	22.5000	24.0000
5× 7	13.1250	14.5833	16.0418	17.5000	18.9583	20.4167	21.8750	23.3333
6× 7	15.7500	17.5000	19.2500	21.0000	22.7500	24.5000	26.2500	28.0000
7× 7	18.3748	20.4167	22.4582	24.5000	26.5416	28.5833	30.6250	32.6667
6× 8	18.0000	20.0000	22.0000	24.0000	26.0000	28.0000	30.0000	32.0000
7× 8	21.0000	23.3333	25.6667	28.0000	30.3333	32.6667	35.0000	37.3333
8× 8	24.0000	26.6667	29.3333	32.0000	34.6667	37.3333	40.0000	42.6667
7× 9	23.6250	26.2500	28.8750	31.5000	34.1250	36.7500	39.3750	42.0000
8× 9	27.0000	30.0000	33.0000	36.0000	39.0000	42.0000	45.0000	48.0000
9× 9	30.3750	33.7500	37.1250	40.5000	43.8750	47.2500	50.6250	54.0000
8×10	30.0000	33.3333	36.6667	40.0000	43.3333	46.6667	50.0000	53.3333
9×10	33.7500	37.5000	41.2500	45.0000	48.7500	52.5000	56.2500	60.0000
10×10	37.5000	41.6667	45.8333	50.0000	54.1667	58.3333	62.5000	66.6667

TIES PER 1,000 FT. AND PER MILE OF TRACK

Length of Track	Distance From Center to Center of Ties, in Inches						
	18	20	22	24	26	28	30
	Number of Ties						
1,000 ft...	667	600	546	540	462	429	400
1 mi......	3,520	3,168	2,880	2,640	2,437	2,267	2,112

EXAMPLE.—How many feet, board measure, are there in the ties required to lay 1,500 ft. of track; the ties are 6 ft. 6 in. long, 5 in.$\times$6 in. in cross-section, and spaced 22 in. between centers?

SOLUTION.—1,500 ft.$=1\frac{1}{2}$ thousands of feet. From the accompanying tables, $1\frac{1}{2}\times546\times16\frac{1}{4}=13{,}308\frac{3}{4}$, say, 13,500 ft., B. M.

THE PREPARATION OF COAL

CRUSHING MACHINERY

The object of crushing ore or coal is: first, to free the mineral or other valuable constituents from the gangue, slate, pyrites (sulphur), or other worthless or objectionable constituents so that they can be subsequently separated; or, second, simply to reduce the size of the individual pieces and so get the material into a more salable or convenient condition for use.

Selection of a Crusher.—The style of crusher employed is influenced by: The amount of material to be crushed in a given time. The size of the material as it goes to the crusher. The physical characteristics of the material to be crushed; that is, whether it is hard or soft, tough or brittle, clayey or sticky. The object of the crushing. The character of the product desired; that is, whether an approximately sized product is desirable and whether dust or fine material is objectionable.

The term *cracking rolls* is applied to rolls having teeth, which are usually made separate and inserted. These rolls, Fig. 1, are used for breaking coal, phosphate rock, etc., the object being to break the material into angular pieces with the smallest possible production of very fine material. The principle field for cracking rolls is in the preparation of anthracite, and the exact style or design of the roll depends largely on the physical condition of the coal under treatment. In most cases, the rolls are constructed with an iron cylinder having steel teeth inserted, the size, spacing, and form of the teeth depending on the size and physical condition of the material to be broken. Cracking

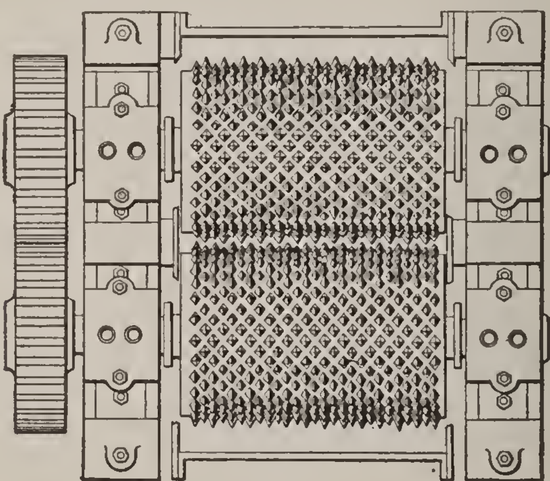


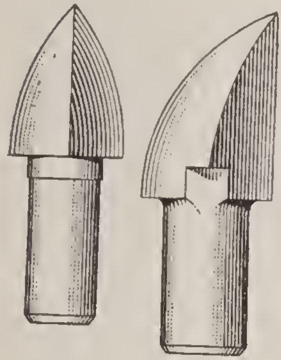
FIG. 1

rolls vary from 12 to 48 in. in diameter and from 24 to 36 in. in surface width. The teeth of the larger sizes are from 3 to 3½ in. high, and of the smaller 1 in. or less.

The form of the teeth varies greatly, but, as a rule, the larger rolls have straight pointed teeth of the sparrow-bill or some similar form, Fig. 2 *a*. The old curved, or hawk-billed, teeth, Fig. 2 *b*, have now gone wholly out of use.

Disintegrating rolls and *pulverizers* are sometimes used to reduce coking coal to the size of corn or rice before introducing it into the ovens. One roll is driven at double the speed of the other, the slower roll acting as a feed roll, and the other as a disintegrator.

For the reduction of coal, crushers employing *hammers* have been used, Fig. 3. The crushing chamber is usually of a circular or barrel form, and the crushing is done by means of hammers pivoted about a central shaft. These swing out by centrifugal force and strike blows upon the coal to be broken.



(a) (b)

FIG. 2

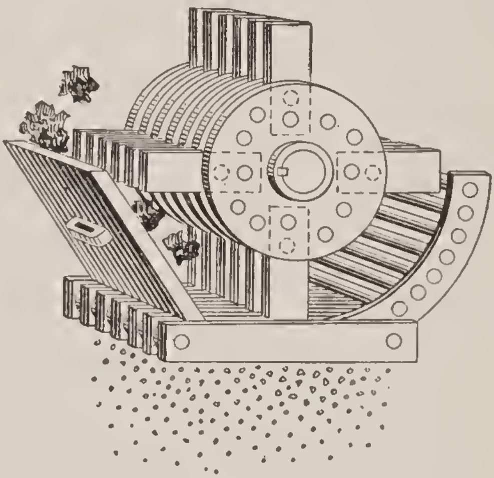


FIG. 3

When it is reduced sufficiently fine, it is discharged through bars or gratings at the lower portion of the machine. This style of machinery is usually employed in preparing coal for coke ovens, thus occupying the same field as the disintegrating rolls.

SIZING AND CLASSIFYING APPARATUS

Platform Bars.—In the anthracite breakers, the terms *platform bars* or *head-bars* are usually employed for the bars that remove the fine material from the run of mine, so that the coarse will go to the crushers. These bars are made of 1½-in. to 2-in. round iron placed at an inclination of 5-in. to 1-ft., the spacing depending on the size of coal it is desired to make in the breaker.

Shaking Screens.—Shaking screens have an advantage in that the entire area of the screen is available for sizing, and hence a greater capacity can be obtained from a given area of screening surface. They also occupy less vertical height than a revolving screen. In coal breakers they are particularly

applicable where the coal is wet and has a tendency to stick together. The principal disadvantage of the shaking screen is that the reciprocating motion imparts a vibration to the framing of the building.

The capacities of shaking screens operating on anthracite have been given as follows. The parties giving these figures advise the use of 140 R. P. M. for the cam-shaft. For broken and egg coal, $\frac{1}{4}$ sq. ft. per T. for 10 hr. For stove and chestnut coal, $\frac{1}{2}$ sq. ft. per T. for 10 hr. For pea and buckwheat coal $\frac{3}{4}$ sq. ft. per T. for 10 hr. For birdseye and rice, $1\frac{1}{4}$ sq. ft. per T. for 10 hr. For sizing bituminous coal, inclined shaking screens are extensively used in certain sections, particularly in the Middle Western States. These screens are given a shaking motion by means of cams and connecting-rods, which make from 60 to 100 strokes per minute, the speed varying according to the amount of moisture in the coal.

Size of Mesh.—The perforations given in the accompanying table have been adopted by two of the largest anthracite companies as the dimensions for the holes in shaking screens to produce sizes equivalent to those produced by revolving screens.

MESH FOR SHAKING SCREENS

Kind of Coal	Lehigh Valley Coal Co.	Phila. & Reading Coal & Iron Co.		Kind of Coal
	Round Inches	Round Inches	Square Inches	
Steamboat . . .		$5\frac{3}{8}$	5	Steamboat
Lump.	$4\frac{1}{2}$	$4\frac{1}{2}$	4	Large broken
Broken.	$3\frac{1}{4}$	$3\frac{1}{4}$	$2\frac{3}{4}$	Small broken
Egg.	$2\frac{5}{16}$	$2\frac{1}{4}$	2	Egg
Stove.	$1\frac{5}{8}$	$1\frac{1}{2}$	$1\frac{3}{8}$	Stove
Chestnut.	$\frac{15}{16}$	$\frac{7}{8}$	$\frac{3}{4}$	Chestnut
Pea.	$\frac{5}{8}$	$\frac{9}{16}$	$\frac{1}{2}$	Pea
Buckwheat. . .	$\frac{11}{32}$	$\frac{5}{16}$	$\frac{1}{4}$	Buckwheat
Rice.	$\frac{3}{16}$	$\frac{3}{16}$	$\frac{1}{8}$	Rice

Revolving Screens, or Trommels.—The screen is placed about the periphery of a cylinder or frustum of a cone. The material to be sized is introduced at one end; the small size passes through the screen, and the other size is discharged from the other end. If the form is cylindrical, the supporting shaft must be inclined so that the material will advance toward the discharge end. The inclination determines the rapidity with which the material will be carried through the screen. The advantage of the conical screen is that the shaft is horizontal and hence the bearings are simpler; this a very decided advantage where the machinery must be crowded into a minimum space, and is hard to get at.

Speed.—The periphery of a revolving screen should travel about 200 ft. per min. In the case of very fine material, screens are sometimes run faster than this. The following have been adopted as standard speeds for screens by one of the largest anthracite companies:

Speed of Screens

	Rev. per Min.		*Rev. per Min.*
Mud-screens..........	8.87	Big screens...........	8.52
Counter mud screens ..	15.49	Pony screens	10.87
Cast-iron screens......	11.25	Buckwheat screens....	15.30

Duty of Anthracite Screens.—The following list gives the number of square feet of screen surface required for a given duty in the case of revolving screens working upon anthracite:

Egg Coal.............	1 T. per 1 sq. ft. per 10 hr.
Stove coal............	1 T. per $1\frac{1}{4}$ sq. ft. per 10 hr.
Chestnut coal.........	1 T. per $1\frac{1}{2}$ sq. ft. per 10 hr.
Pea coal..............	1 T. per 2 sq. ft. per 10 hr.
Buckwheat coal	1 T. per $2\frac{3}{4}$ sq. ft. per 10 hr.
Rice coal	1 T. per $3\frac{3}{4}$ sq. ft. per 10 hr.
Culm	1 T. per 5 sq. ft. per 10 hr.

These figures may be reduced from 20% to 30% for very dry or wash coal.

Revolving Screen Mesh for Anthracite.—A standard mesh for revolving screens for sizing anthracite was adopted some years ago, but it is only approximately adhered to and a considerable variation from the standard is found throughout the anthracite region. The following are probably as nearly standard meshes for revolving screens for sizing anthracite coal as can be given:

MESH FOR SIZING COAL

Culm	passes through $\frac{3}{32}$-in. mesh
Birdseye	passes over $\frac{1}{4}$-in. mesh, and through $\frac{5}{16}$-in. mesh
Buckwheat	passes over $\frac{1}{4}$-in. mesh, and through $\frac{1}{2}$-in. mesh
Pea	passes over $\frac{1}{2}$-in. mesh, and through $\frac{3}{4}$-in. mesh
Chestnut	passes over $\frac{3}{4}$-in. mesh, and through $1\frac{3}{8}$-in. mesh
Stove	passes over $1\frac{3}{8}$-in. mesh, and through 2-in. mesh
Egg	passes over 2-in. mesh, and through $2\frac{3}{4}$-in. mesh
Grate	passes over $2\frac{3}{4}$-in. mesh, and out end of screen
Special grate	passes over 3-in. mesh, and out end of screen
Special steamboat	passes over 3-in. bars, and through 6-in bars.

Jigs.—The general term *jig* is applied to that class of concentrating machines in which the separation of the mineral

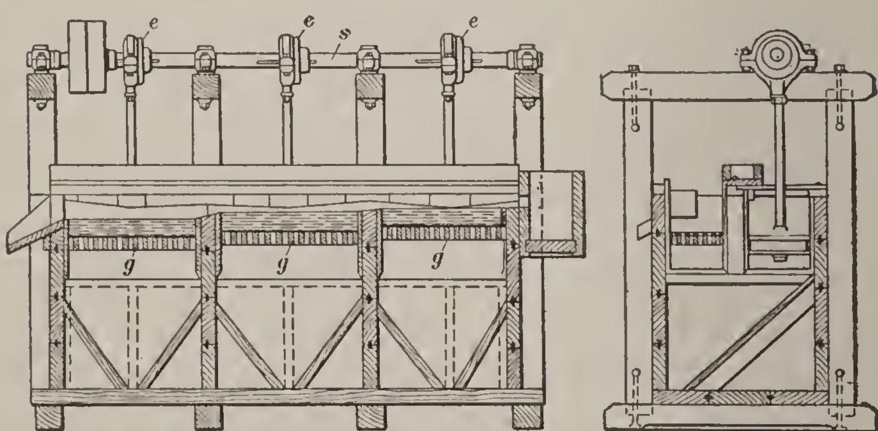


FIG. 4

from the gangue takes place on a screen or bed of material and is effected by pulsating up-and-down currents of a fluid medium. A number of different methods are used for driving the pistons that cause the pulsations of the water in jigs. Some use plain eccentrics, giving the same time to both the up and the down strokes of the pistons, while others employ special arrangements

of parts, which give a quick down stroke and a slow up stroke, thus allowing the water ample time to work its way back through the bed without any sucking action from the piston. This tends to make a better separation in some cases than the use of the plain eccentrics.

Stationary screen jigs are illustrated by Fig. 4, which shows a 3-compartment jig. The separation takes place on screens supported on wooden frames *g*, and is effected by moving the water in each compartment so that it ascends through the screen, lifting the mineral and allowing it to settle again, thus giving the material an opportunity to arrange itself according to the law of equally falling particles.

Removal of Sulphur from Coal.—The object of washing coal is to remove the slate and pyrites, thus reducing the amount of ash and sulphur. Many forms of washers easily and cheaply reduce the slate from 20% in the coal to 8% of ash in the coke, but it is much more difficult to reduce 4% of sulphur in the coal to 1% or less of sulphur in the coke. Sulphur occurs in the coal in three forms, as hydrogen sulphide, calcium sulphate, and pyrite.

HANDLING OF MATERIAL

Anthracite.—The following may be taken as average figures for the angle or grade of chutes for anthracite, to be used where the chutes are lined with sheet steel: For broken or egg coal, $2\frac{1}{2}$ in. per ft.; for stove or chestnut coal, $3\frac{1}{2}$ in. per ft.; for pea coal, $4\frac{1}{4}$ in. per ft.; for buckwheat coal, 6 in. per ft.; for rice coal, 7 in. per ft.; for culm, 8 in. per ft.

If the coal is to start on the chute, 1 in. per ft. should be added to each of the foregoing figures; while if the chutes are lined with manganese bronze in place of steel, the figures can be reduced 1 in. per ft. for coal in motion, or would remain as in the table to start the coal. When the run of mine is to be handled, as in the main chute, at the head of the breaker, the angle should be not less than 5 in. per ft., or practically $22\frac{1}{2}°$ from the horizontal. If chutes for hard coal are lined with glass, the angle can be reduced from 30% to 50%, depending somewhat on the nature of the coal. In all cases, the flatter the coal, the steeper the angle must be, on account of the large friction surfaces exposed, compared with the weight of the piece. If the

chutes are lined with cast iron, the angle should be about the same as that employed for steel, though sometimes a slightly greater angle is allowed.

The accompanying table is printed through the courtesy of the Link-Belt Engineering Co., Philadelphia, Pa.:

PITCH AT WHICH ANTHRACITE WILL RUN

Size of Coal	Dry Coal					Wet Coal	
	Sheet-Iron Lining		Cast-Iron Lining	Glass Lining			
	Pitch of Chute, in Inches per Foot						
	Start On	Continue On	Start On	Start On	Continue On	Start On	Continue On
Broken slate.....	$5\frac{1}{4}$	$4\frac{3}{8}$	$5\frac{3}{8}$	$3\frac{1}{4}$	3		
Dry egg slate....	$5\frac{1}{4}$	$4\frac{3}{8}$	$5\frac{1}{4}$	$3\frac{1}{4}$	3		
Dry stove slate...	$5\frac{1}{4}$	$4\frac{3}{8}$	$5\frac{1}{4}$	$3\frac{1}{4}$	3		
Dry chestnut slate	5	$4\frac{3}{8}$	$5\frac{1}{4}$	$3\frac{1}{4}$	3		
Broken coal......				$2\frac{5}{8}$	$2\frac{1}{4}$		
Egg coal.........	$3\frac{3}{8}$	3	$3\frac{7}{8}$	$2\frac{5}{8}$	$2\frac{1}{4}$	$2\frac{1}{4}$	$1\frac{3}{4}$
Stove coal.......	$4\frac{3}{4}$	$4\frac{1}{4}$	$4\frac{3}{4}$	3	$2\frac{1}{2}$	$2\frac{1}{2}$	$2\frac{1}{8}$
Chestnut coal....	$4\frac{3}{4}$	$4\frac{1}{4}$	$4\frac{3}{4}$	3	$2\frac{1}{2}$	3	$2\frac{1}{2}$
Pea coal.........	$5\frac{1}{2}$	5	$5\frac{3}{4}$	$3\frac{1}{4}$	$2\frac{1}{2}$	3	$2\frac{9}{16}$
Buckwheat No. 1.				$3\frac{5}{8}$	$3\frac{1}{8}$	$3\frac{1}{2}$	$3\frac{1}{8}$
Buckwheat No. 2.				$3\frac{3}{4}$	$3\frac{1}{4}$	$3\frac{3}{4}$	$3\frac{1}{4}$
Buckwheat No. 3.				$4\frac{3}{8}$	$3\frac{3}{8}$	$4\frac{7}{8}$	$4\frac{1}{8}$
Buckwheat No. 4.				$4\frac{7}{8}$	$4\frac{1}{8}$	$4\frac{7}{8}$	$4\frac{1}{8}$

Bituminous Coal.—When the run of mine is to be handled, the angle of the chutes should be from 35° to 45° from the horizontal, or from $8\frac{1}{2}$ in. to 12 in. per ft. If the coal is wet, the angle should always be steeper, as coarse coal will slide on a flatter angle than slack or fine coal.

BRIQUETING

Fuel, fuel dust, and other products may be briqueted by a number of different styles of machines, but all these may be divided into two classes, *briquet* and *eggette machines.* Fuel briquets have not come into general use in the United States on account of the great amount of cheap fuel available, which has prevented the utilization of culm, coal dust, etc.; and on account of the lack of or high price of suitable bonding material. This latter condition is now being removed by the introduction of by-product coke ovens, from which supplies of coal tar can be obtained.

SPACE OCCUPIED BY 2,000 LB. OF VARIOUS COALS

Anthracite	Broken Cubic Feet	Egg Cubic Feet	Stove Cubic Feet	Chestnut Cubic Feet	Pea Cubic Feet
Lackawanna	37.10	36.65	34.90	34.35	37.25
Garfield red ash	37.30	36.95	36.35	36.35	37.50
Lykens Valley	37.55	37.25	37.55	37.25	38.50
Shamokin	38.05	37.70	37.25	37.25	38.50
Plymouth red ash	34.90	34.85	34.75	34.70	36.90
Wilkes-Barre	34.95	34.35	33.75	34.00	36.90
Lehigh	33.30	33.80	33.55	32.55	33.05
Lorberry	34.65	34.20	33.80	33.55	35.20
Scranton	35.35	35.20	34.60	33.30	34.95
Pittston	35.45	34.95	34.35	33.70	35.50

Bituminous	Cubic Feet	Bituminous	Cubic Feet
Cumberland	36.65	Pocahontas	34.00
Clearfield	33.55	American cannel	41.50
New River	40.15	English cannel	42.30

TREATMENT OF INJURED PERSONS

The dangers to be feared in case of wounds, are shock or collapse, loss of blood, and unnecessary suffering in the moving of the patient.

In *shock*, the injured person lies pale, faint, and cold, sometimes insensible, with feeble pulse and superficial breathing. The cause of death in case of a shock is arrest of heart action, produced by the suspension of the functions of the brain and spinal cord. In treatment, the two most important parts are: the position of the injured person and the application of external warmth.

The injured person should at once be placed in a recumbent position, his head resting on a plane lower than that of his trunk, legs, and feet. He should be well wrapped up and protected from the chilling influences of external air. Where there is danger of immediate death, stimulants should be given; in all other conditions of shock, stimulants are injurious.

Loss of Blood.—In case of loss of blood, two conditions present themselves: (1) The bleeding is arrested spontaneously or otherwise, but the injured person presents all the symptoms of loss of blood; (2) the injured person is actually bleeding, and he is, or is not, suffering from loss of blood.

In the first condition, life is threatened by anemia of the brain and spinal cord, and all the efforts of treatment should be to direct the flow of whatever quantity of blood that may still remain in the body to these vital centers. This is most efficiently done by placing the injured person in a recumbent position, with his head resting on a plane somewhat lower than that of his trunk and legs. In graver cases, constricting bands should be applied to both arms, as near the shoulders as possible, and to both thighs, as near the abdomen as possible. This last maneuver directs the entire quantity of blood in the body to the suffering centers, the centers of life itself. Stimulants may be sparingly administered.

If there is bleeding, do not try to stop it by binding up the wound. The current of blood to the part must be checked. To do this, find the artery, by its beating; lay a firm and even compress or pad (made of cloth or rags rolled up, or a round

stone or piece of wood well wrapped) over the artery, as shown in Fig. 1. Tie a handkerchief around the limb and compress; put a bit of stick through the handkerchief and twist the latter

FIG. 1

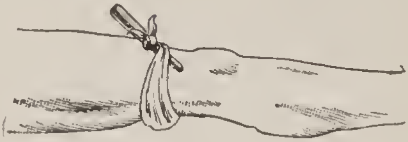


FIG. 2

up until it is just tight enough to stop the bleeding; then put one end of the stick under the handkerchief, to prevent untwisting, as in Fig. 2.

The artery in the thigh runs along the inner side of the muscle in front of the bone, as shown by dotted line in Fig. 3. A little above the knee it passes to the back of the bone. In injuries at or above the knee, apply the compress higher up, on the inner side of the thigh, at the point *P*, Fig. 3, with the knot on the outside of the thigh. When the leg is injured below the knee, apply the compress at the back of the thigh, just above the knee, at *P*, Fig. 4, and the knot in front, as in Figs. 1 and 2.

The artery in the arm runs down the inner side of the large muscle in front, quite close to the bone, as shown by dotted line; low down it is further forwards, toward the bend of the elbow. It is most easily compressed a little above the middle, at *P*, Fig. 5. Care should be taken to examine the limb from time to time, and to lessen the compression if it becomes cold or purple; tighten up the handkerchief again if the bleeding begins afresh.

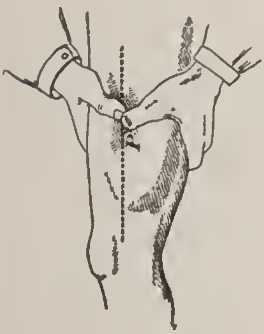


FIG. 3

FIG. 4

To Transport a Wounded Person Comfortably.—Make a soft and even bed for the injured part, of straw, folded blankets, quilts, or pillows, laid on a board with side pieces of board nailed on, when this can be done. If possible, let the patient

be laid on a door, shutter, settee, or some firm support properly covered. Have sufficient force to lift him steadily, and let those that bear him not keep step.

Fig. 5

Should any important arteries be opened, apply the handkerchief, as recommended. Secure the vessel by a surgeon's dressing forceps, or by a hook, then have a silk ligature put around the vessel, and tighten. Should the bleeding be from arterial vessels of small size, apply persulphate of iron, either in tincture or in powder, by wetting a piece of lint or sponge with the solution; then, after bleeding ceases, apply a compress against the parts, to sustain them during the application of the persulphate of iron, and to prevent further bleeding, should it occur. The persulphate of iron should be kept in or about all working places.

Fig. 6

Bleeding From Scalp Wounds.—A pad or compress is placed immediately before the ear, over the region marked by a dotted line, Fig. 6. The compress is firmly secured by a handkerchief. If this does not arrest bleeding, a similar compress on the opposite side should be applied. Should the bleeding issue from a wound of the posterior or back part of the head, a compress should be placed behind the ear, over the region marked by the dotted line, Fig. 6, and firmly secured by a handkerchief or bandage.

TREATMENT OF PERSONS OVERCOME BY GAS

Miners are exposed to asphyxia when the circulation of the air is not sufficiently active, when the mine exhales a quantity of deleterious gas, when they penetrate into old and abandoned workings, and when there is an explosion. The symptoms of asphyxia are sudden cessation of the respiration, of the

pulsations of the heart, and of the action of the senses; the countenance is swollen and marked with reddish spots, the eyes are protruded, the features are distorted, and the face is often livid, etc. The best and first remedy to employ, and in which the greatest confidence ought to be placed, is the renewal of the air necessary for respiration. Proceed as follows:

1. Promptly withdraw the asphyxiated person from the deleterious place and expose him to pure air.

2. Loosen the clothes round the neck and chest, and dash cold water in the face and on the chest.

3. Attempts should be made to irritate the mucous membrane with the feathered end of a quill, which should be gently moved in the nostrils of the insensible person, or to stimulate it with a bottle of volatile alkali placed under the nose.

4. Keep up the warmth of the body, and apply mustard plasters over the heart and around the ankles.

5. If these means fail to produce respiration, Docter Sylvester's method of producing artificial respiration should be tried as follows: Place the patient on the back on a flat surface, inclined a little upwards from the feet; raise and support the head and shoulders on a small firm cushion or folded article of dress placed under the shoulder blades. Draw forwards the patient's tongue and keep it projecting beyond the lips; an elastic band over the tongue and under the chin will answer this purpose, or a piece of string or tape may be tied around them, or by raising the lower jaw the teeth may be made to retain the tongue in that position. Remove all tight clothing from about the neck and chest, especially the suspenders. Then standing at the patient's head, grasp the arms just above the elbows, and draw the arms gently and steadily upwards above the head, and keep them stretched upwards for 2 sec. (by this means air is drawn into the lungs). Then turn down the patient's arms and press them gently and firmly for 2 sec. against the sides of the chest (by this means air is pressed out of the lungs). Repeat these measures alternately, deliberately, and perseveringly about 15 times in a minute, until a spontaneous effort to respire is perceived, immediately upon which cease to imitate the movements of breathing, and proceed to induce circulation and warmth.

6. To promote warmth and circulation, rub the limbs upwards with firm, grasping pressure and energy, using handkerchiefs, flannels, etc. Apply hot flannels, bottles of hot water, heated bricks, etc., to the pit of the stomach, the arm pits, between the thighs, and to the soles of the feet.

7. On the restoration of life, a teaspoonful of warm water should be given, and then, if the power of swallowing has returned, small quantities of wine, warm brandy and water, or coffee should be administered.

8. These remedies should be promptly applied, and as death does not certainly appear for a long time, they ought only to be discontinued when it is clearly confirmed. Absence of the pulsation of the heart is not a sure sign of death, neither is the want of respiration.

SEE FOLLOWING PAGES

General Superintendent Over 1,000 Men

When I enrolled in the Complete Coal Mining Course of the International Correspondence Schools, my education was confined to a knowledge of how to read and write. Notwithstanding the disadvantage of so poor an education, your instruction carried me through and I passed a creditable examination. I consider that it is to your Schools that I owe my advancement. When I enrolled I was a mine boss; I am now General Superintendent for the National and Parkdale Fuel Companies, of Denver, Colo., and my salary has been increased 125 per cent. There never was a time in the history of the United States when good, competent, and reliable mine foremen and superintendents were so much in demand as at present; and any intelligent mine worker who has the ambition can fit himself to assume the responsibility by taking a Course in Mining with the I. C. S. and completing it in leisure time that could not be spent in anything more advantageous to himself.

JOS. WATSON,
Louisville, Colo.

BEGAN WORKING WHEN 8 YEARS OLD

J. M. Baker, Woodland, Pa., had received but an imperfect education when he took up our Complete Coal Mining Course, having begun to work in the mine at the age of 8. At the time of his enrolment his wages were so small that he could with difficulty support himself. His Course has been of the greatest benefit to him, enabling him to become mine superintendent for the Harbison-Walker Refractories Company, having eight foremen and several hundred men at work under his direction. His wages have been increased 500 per cent.

BECAME GENERAL MANAGER

Wm. Hollis, Cordova, Ala., says that it was his Complete Coal Mining Course with the I. C. S. which enabled him to rise from the position of mine foreman to that of general manager of the Alberta Coal, Mineral and Lumber Company. His salary has been doubled since he enrolled with us.

DRAWS $300 A MONTH

While working as top boss of a small mine, Gus Blair, Murphysboro, Ill., enrolled with the I. C. S. for the Complete Coal Mining Course. As he had worked in the mines from the age of 9, it was hard work for him to confine himself to study. However, he pursued the Course until his education was improved, and made a substantial advancement in position and salary. At the time of enrolment he was paid $50 a month. He now draws $300 a month from the Gus Blair Big Muddy Coal Company, of which he is half owner and general manager.

A GRADUATE'S SUCCESS

S. J. Routledge, Kellerman, Ala., holds I. C. S. Diplomas both in Coal Mining and in Surveying and Mapping. When he enrolled for the first Course he was earning about $75 a month. He is now drawing a salary 150 per cent. larger as the superintendent of coal mines for the Central Iron and Coal Company. He says that his present position is due solely to the knowledge gained from his I. C. S. Course.

EMPLOYS 500 MEN

Chas. A. Sine, Johnson City, Ill., left school before he knew the multiplication table. When he enrolled for our Coal Mining Course, he was driving a mule, earning $40 a month. Before he received his diploma he was able to pass the examination for mine manager. He is now superintendent of the Johnson City Coal Company, employing 500 men.

GRADUATE BECOMES SUPERINTENDENT

S. B. Isenburg, Osceola Mills, Pa., was working as a laborer when he enrolled with the I. C. S. for the Short Coal Mining Course, from which he graduated. This enabled him to pass the state examination for mine foreman and to become the mine superintendent of the Blair Brothers Coal Company, with an increase in salary of 150 per cent.

Income Ten Times As Large

I used to feel that I was working hard enough without having to devote my nights to study, when employed as a clerk at $45 a month in the mining department of the Cambria Steel Company. However, I stuck to the Complete Coal Mining Course, which enabled me to gain a first-class certificate of competency as fire boss, and afterwards as mine foreman. I am at present superintendent of the Johnstown mines of the Cambria Steel Company, employing 2,000 men. In addition, I am a stockholder and director in several other companies, consequently my income is at least 10 times what it was when I enrolled.

Geo. T. Robinson,
143 Green St., Johnstown, Pa.

NOW PROPRIETOR

James Nevin, Ottumwa, Iowa, while working as a hoisting engineer enrolled with the I. C. S. for the Complete Coal Mining Course. He gives the Schools the highest indorsement, because they have enabled him to become superintendent of the Trio Coal Company, of which he is also part owner.

ONCE A MULE DRIVER

While driving a mule in the mines at the age of 19, John Clapperton, Jr., Minden, W. Va., enrolled for the Complete Coal Mining Course. Through the knowledge he obtained from this he has been able to pass two examinations and to become superintendent of the New River Coal Company, largely increasing his salary thereby.

A GOOD FRIEND OF THE SCHOOLS

A good friend of the Schools, Jos. Knapper, Philipsburg, Pa., advises his friends to study I. C. S. Courses, because of his experience since enrolment for our Complete Coal Mining Course. Mr. Knapper was earning $75 a month at the time of enrolment. After pursuing his Course he rose step by step until he is now state mine inspector for the eighth district, at a salary of $3,000 a year.

CREDITS HIS SUCCESS TO THE I. C. S.

D. J. Griffith, Trinidad, Colo., was earning only $20 a month as a miner, at the age of 37, when he enrolled with us for the Complete Coal Mining Course. After graduation he served the state of Colorado for a time as chief inspector of coal mines. He is now chief inspector of mines for the American-Victor Fuel Company, and he attributes all his success to the I. C. S.

PAY INCREASED 233 PER CENT.

P. J. Moore, Carbondale, Pa., was employed as a fire boss when he enrolled for the Complete Coal Mining Course. He is now state mine inspector for the first anthracite district, and pay days bring him 233 per cent. more than they did at the time of enrolment.

DIRECTS 10,000 MEN

W. R. Calverley, Windber, Pa., was a miner when he enrolled for the Complete Coal Mining Course. By diligent study he advanced to the position of general superintendent of the Berwind-White Coal Mining Company, having the welfare of 10,000 men committed to his charge. He has always given great credit to the Schools.

Now State Mine Inspector

When I enrolled in the Complete Coal Mining Course of the International Correspondence Schools, Scranton, Pa., I had had about 20 months of schooling all told. I was employed at the time as assistant foreman, and was getting $55 a month. After enrolling in the Schools, I was soon advanced to the position of mine foreman at $75 a month, which was voluntarily increased to $90 a month. I am now Mine Inspector of the Fifth Bituminous District, at a salary of $3,000 a year.

Correspondence instruction, as conducted by the I. C. S., is the finest and most complete in the world today; every young man that desires to advance or better his condition should enroll at once. No one can enroll with you and apply himself to his work, without being greatly benefited.

I shall be glad to answer any inquiries regarding the Schools and my Course with them.

Isaac G. Roby,
Inspector, Fifth Bituminous District, Uniontown, Pa.

STATE MINE INSPECTOR

By diligent study of the Complete Coal Mining Course, for which he enrolled with the I. C. S., Jos. Williams, 245 Beale Ave., Altoona, Pa., has risen from a position as miner to that of inspector of mines, for the tenth bituminous district, State of Pennsylvania. His salary has increased from $45 a month to $3,000 a year, and he gives the I. C. S. all the credit.

NOW MANAGER

T. E. Moore, Eyremore, Alta., Can., was working as a shift man for the Prairie Coal Company, when he enrolled with the Schools for the Complete Coal Mining Course. He is still working for the same company, and he has risen to the position of manager of their mine on the Bow River at a salary of $150 a month.

COULD HARDLY WRITE HIS NAME

A. W. Courtney, Princeton, B. C., was 25 years old and was working as a laborer when he enrolled for the Short Coal Mining Course. At the time he could hardly write his name. Keeping diligently at his studies, he was able to pass the examination for mine foreman and now holds a foreman's position at a salary of $150 a month.

THREE TIMES HIS FORMER SALARY

J. J. Clark, Sagamore, Pa., began to reap the benefits of his study on the Complete Coal Mining Course 10 months after enrolment. By devoting all his spare time to his Course, he was able to obtain a mine foreman's certificate. He is now assistant superintendent for the Buffalo & Susquehanna Coal and Coke Company, and his wages are three times as great as when he was loading coal.

IN CHARGE OF A LARGE PLANT

When H. L. Fisher, Kayford, W. Va., enrolled with the I. C. S. for the Complete Coal Mining Course his knowledge of mining was very limited. By diligent study of his Course he is now able to take charge of the largest plant of the Cabin Creek Consolidated Coal Company, which includes eight of their principal mines. His salary is $190 a month.

NOW SERVES THE STATE

F. J. Pearce, Rm. 120, State Capitol, Indianapolis, Ind., enrolled with the I. C. S. 12 years ago for the Complete Coal Mining Course. He has now risen to the highest position in his profession, deputy inspector of mines and mining for the State of Indiana, at a salary of $2,000 a year. He had little education before enrolment, but the secret of his advancement lies in the fact that he has used his spare time and his I. C. S. Course to obtain an education.

$540 to $3,000 a Year

In reply to your letter, I beg leave to state that my present position is Mine Inspector, employed by the State of Pennsylvania. When I enrolled in the Full Mining (now the Mining Engineering) Course, my salary was $45 a month. I was employed as bratticeman, at the Woodward Mines. After studying some time, I passed the examination for mine foreman and was appointed assistant mine foreman—later foreman. Then I passed the Mine Inspector's examination and was elected to that position. My present salary is $250 a month. I can conscientiously recommend the International Correspondence Schools to any young man that has any desire to advance himself.

L. M. Evans,
Inspector Second Anthracite Inspection District, 10 Belmont Terrace, Scranton, Pa.

NOW A MINE OWNER

J. P. Davis, Columbia, Mo., was earning $75 a month as mine foreman when he enrolled with the I. C. S. for the Complete Coal Mining Course. This has been so profitable to him that he is now senior partner and manager of the Davis & Watson Coal Company, employing 30 men.

NOW A FOREMAN

Howell John, Box 48, Meritt, B. C., declares that his I. C. S. Mine Foremen's Course advanced him to the position of foreman with the Pacific Coast Collieries. Mr. John began working in the mines at 13 years of age and was a miner when he enrolled. His present position pays him $135 a month

EARNS $200 A MONTH

While working as a timberman, John Prentice, Lundbreck, Alta., Can., took up the Mine Foremen's Course with the I. C. S. At the time he was earning $70 a month. He is now mine manager for the Breckenridge & Lund Coal Company, earning $200 a month, and he says it was the I. C. S. that made the difference.

WORTH $40 A MONTH TO HIM

His Mine Foremen's Course with the I. C. S., for which Addison Shaw, Berryburg, W. Va., subscribed, was the means of advancing him to the position of mine foreman for the Consolidated Coal Company, with an increase in salary of $40 a month.

NOW AN OFFICER OF THE COMPANY

R. S. Burchinal, Smithfield, Pa., says that the Mine Foremen's Course for which he enrolled with the I. C. S., was the cause of his obtaining a foreman's certificate which enables him to look after the mine, as well as the outside management of his company. He is now treasurer and general manager of the Smithfield Coal and Coke Company, receiving a salary of $125 a month.

A WORLD OF GOOD

The experience of Gus Champ, Cherokee, Kans., shows what the I. C. S. Mine Foremen's Course will do for the man that has the grit to go ahead. Mr. Champ says that his Course did him a world of good, since it has advanced him to the position of foreman for the Hamilton Coal Company, at a salary of $100 a month.

Earning $3,000 a Year

HOWARD M. BLACK,
Mining Engineer, Grass Valley, Cal.

International Correspondence Schools,
Scranton, Pa.

GENTLEMEN:—At the time of enrolment as a student of the I. C. S., I was superintending a small mine at a salary of $100 a month. I have finished my Metal Mining Course, with the exception of geometrical drawing. I still hold the same position, but as this only occupies part of my time, I make a specialty of examining and reporting on mines, and do considerable assaying and other work in the line of mining engineering. For this outside work I receive $12 a day and expenses. I can safely say that my income since enrolment has been increased from the original $1,200 a year to $3,000 a year, due in great part to the technical knowledge acquired through the I. C. S. Course.

Very truly yours,

HOWARD M. BLACK,
Grass Valley, Cal.

NOW PRESIDENT

G. W. Wilmott, Maryd, Pa., was earning about $60 a month as chief repairman about the mines, when he enrolled for the Mine Mechanical Course. After rising to the position of superintendent, in charge of 450 men, he resigned to become president and general manager of the Wilmott Engineering Company, his present position.

LARGEST OF ITS KIND

E. E. Carter, Quartzburg, Idaho, commenced studying the Complete Metal Mining Course soon after leaving grammar school, while working for $10 a week. Later he enrolled for the Complete Metallurgy Course. He considers that his success is largely due to the instruction he received from the I. C. S. At present he is manager of the largest coal mines in Idaho.

FROM $2.50 A DAY TO $38.20 A WEEK

Henry Hoard, Selwood, Ore., feels that he owes to the I. C. S. all his success. He was working around the mines as a mucker, or at anything else he could get, when he enrolled for the Metal Mining Course. His wages then averaged $2.50 a day. He is now assistant foreman for the John Clark Lead Company, employing some 40 men, and his salary averages $38.20 a week.

BECAME SUPERINTENDENT

E. A. Roberts, Entwistle, Alta., Can., was working as a steam engineer for $80 a month when he enrolled with the Schools for the Mining Engineering Course. He is now manager of the shaft sinking work for the Pembina Coal Company, at a salary of $150 a month.

EARNS $160 A MONTH

Geo. H. Shepherd, National, Nev., was earning $2 a day as a millman and sampler, when he enrolled with the I. C. S. for the Metal Mining Course. He now handles the retorting of the huge mass of bullion dispatched from the camp, earning $160 a month.

WHAT THE SCHOOLS DID FOR HIM

In 1909, Peter Kasavage, Johnson City, Ill., could neither read nor write and was earning $2.42 a day as a tracklayer. He then enrolled with the I. C. S. for the Mine Foremen's Course. In July, 1911, he passed the state examination for mine examiner and the next day was appointed to the position of mine examiner of the Illinois Hocking Washed Coal Company, at Marion, Ill.

A Young Man's Success

When I began with the Desoto Coal Mining Development Company I counted cap boards. The president advised me to study Mining Engineering through your Schools. As a result of this study promotion and advancement have been my lot, together with commensurate compensation. My salary has increased 400 per cent. since taking up instruction by mail. If one received the mental training alone, the Course would be worth many times its cost. Today I am Secretary and General Manager of the company and a director and stockholder. To have gone step by step from a counter of cap boards, to the Secretary and General Manager's chair has meant hours and nights and weeks and months of study as well as close application to my duty.

JAS. A. WORSHAM,
Morris, Ala.

SALARY DOUBLED

Nels Johnson, Zeigler, Ill., was earning $60 a month when he enrolled with the I. C. S. for the Short Coal Mining Course. He afterward graduated from the Full Mining Course. He is now mine manager of the Bell & Zoller Mining Company's plant, and his salary has been doubled.

EIGHT TIMES HIS FORMER SALARY

Henry Sankey, Box 756, Cobalt, Ont., was working on a farm at $20 a month when he took up our Metal Mining Course. This has enabled him to become superintendent of the Peterson Lake Mining Company, at a salary of $160 a month.

SIX TIMES HIS FORMER SALARY

E. F. Buffat, Briceville, Tenn., was supporting a family on the small salary of a bookkeeper when he enrolled with the I. C. S. for the Metal Mining Course. He is now superintendent of the Tennessee Coal Company's mine at Briceville, employing about 200 men, at a salary six times what he received at the time of enrolment.

EARNS $250 A MONTH

Lewis R. Smith, 314 Second Ave., Rome, Ga., was an assistant chemist, earning $45 a month when he enrolled with the Schools. His Course in Metallurgy has enabled him to become superintendent of the Silver Creek Furnace Company at a salary of $250 a month.

NOW SUPERINTENDENT

J. W. Powell, Taber, Alta., was working as a fire boss, when he enrolled with the Schools for the Complete Coal Mining Course. Through the study of this Course he qualified himself for a first-class certificate in the Pennsylvania anthracite region, and afterward for the mine foreman's position in Alberta. He is now superintendent of mines for the Canada West Coal Company, Ltd.

WORKING AGAINST ODDS

D. R. Jones, Parrot, Va., had attended school only a few months when he enrolled for our Complete Coal Mining Course. In spite of obstacles he has advanced step by step until he is now superintendent of the Pulaski Anthracite Coal Company, at a salary of $150 a month.

Increased His Salary 500 Per Cent.

I had learned the machinist's trade and was prepared to enter Cornell University, when I enrolled for your Coal Mining Course. I had no knowledge of mine engineering or metal mining at the time, but had worked in the coal mines. In making the radical change from the drift coal mines to the comparatively deep lead mines of this section, I found that my knowledge of metal mining was thorough. I was able to make my work very successful from the start. I found my Course very useful in all my work, and have always been a firm advocate of the I. C. S. At the time of enrolment I was earning $600 a year. I am now general superintendent of the Washburn Lignite Coal Company, of this place, employing 300 men, and my salary has increased 500 per cent.

A. W. Pollock,
Wilton, N. Dak.

NOW MANAGER

Robt. Elminston, Box 696, Panama, Ill., had to start life with a common school education; but with the help of our Short Coal Mining Course, he was able to obtain a first-class certificate in the State of Illinois. This advanced him from a position as miner to that of manager of the Shoal Creek Coal Company's mine No. 1.

NOW SUPERINTENDENT

Geo. E. Loughner, R. F. D. No. 4, Johnstown, Pa., went into the mines at 11 years of age to help support a large family. While he was earning about $50 a month he enrolled for our Short Coal Mining Course, and afterwards for the Complete Coal Mining Course. He is superintendent for the Kelso Smokeless Coal Company, employing 125 men, and his salary has been increased $75 a month.

PASSED WITH 100 PER CENT.

While John Sanderson, Red Lodge, Mont., was working as a miner, he enrolled with the I. C. S. for the Short Coal Mining Course. When he came up for examination he was able to pass with a percentage of 100, although his previous education had been greatly limited, owing to the fact that he began to work in the mines at 10 years of age. He is now acting as foreman for the Northwestern Improvement Company, and his salary has increased 125 per cent.

BECAME FOREMAN

Wm. Fleming, Windber, Pa., was working in the coal mines when he enro led with the I. C. S. for the Short Coal Mining Course. What he learned enabled him to obtain a first-grade mine foreman's certificate, and he now holds the position as foreman of the Eureka No. 42 Mine of the Berwind-White Coal Mining Company. His salary has been increased 60 per cent. since enrolment.

INCREASED HIS SALARY

Because he had studied the Short Coal Mining Course for which he enrolled with the I. C. S., John H. Hauser, Marguerite, Pa., was able to advance from the position of driver to that of mine foreman, increasing his salary to $135 a month.

MINER BECAME SUPERINTENDENT

J. C. Glancy, Pineville, Ky., was working as a miner when he subscribed for the Mine Foremen's Course. Two years ago he secured a state certificate and is now holding a position as mine superintendent for the Pioneer Coal Company, at a salary of $100 a month. He says that the I. C. S. have done wonderful things for him.

State Mine Inspector Salary $3,000

Since I had started to work in the mines when only 9 years old, my education at the time when I enrolled with the International Correspondence Schools for the Short Coal Mining Course, was principally what I had picked up by observation in the school of experience. Without question, your Course offered the best advantage I had ever had, and the farther I went with it the better I liked it. I was so pleased with your treatment of your scholars that I have recommended the I. C. S. to other miners who are now holding positions as foremen and assistant foremen in my district, which should be sufficient guarantee that I have every faith in the I. C. S. My work with the Schools was my best preparation to stand for an examination for State Inspector of Mines, which, I am pleased to say, I was successful in passing. I am now Inspector for the Seventeenth Anthracite Inspection District. When I took up your Course I was receiving a salary of $60 a month. My present salary is $3,000 a year; therefore, you see I have every reason to praise the bridge that carried me over.

I recommend your Course to any one, young or old, who has ambition. It matters not what his previous schooling has been, the I. C. S. will see him through.

Isaac M. Davies,
Lansford, Pa.